GROUND CONTROL RAMIFICATIONS AND ECONOMIC IMPACT OF RETREAT MINING ON ROOM AND PILLAR COAL MINES

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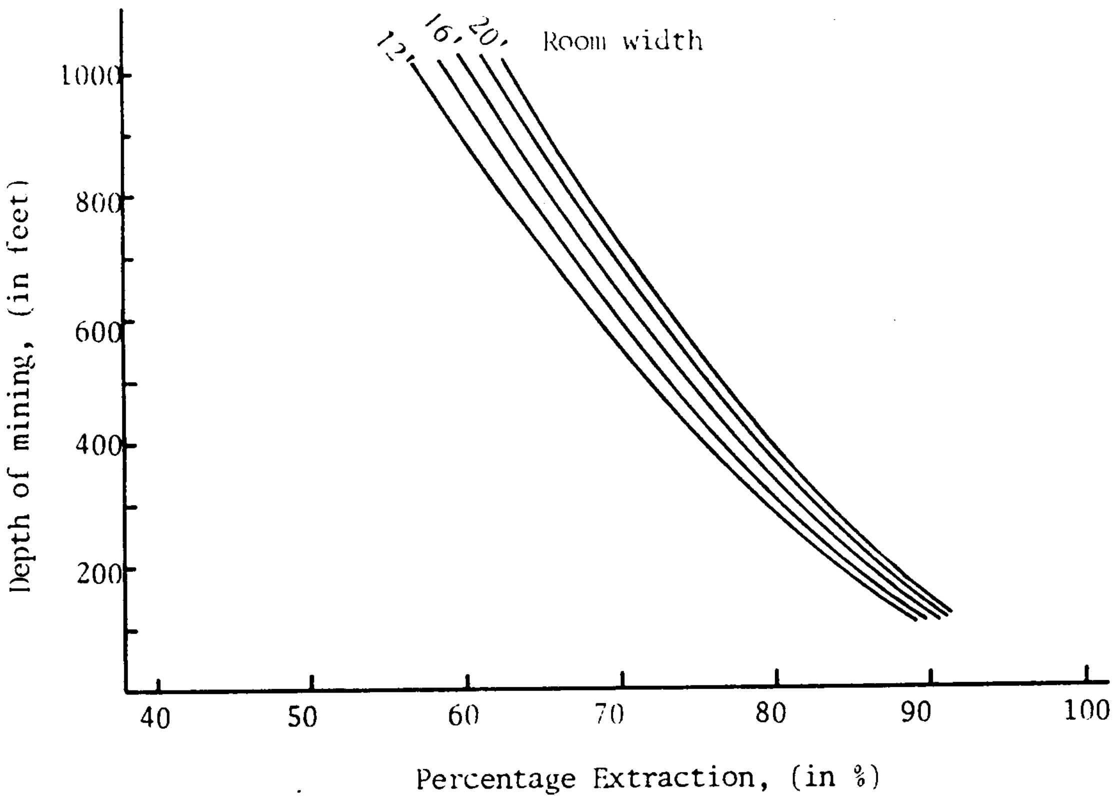
1. INTRODUCTION

Room-and-pillar mining, resulting in partial extraction, has been used as the major underground mining method in this country for more than 100 years. The method is flexible, has a high productive capability and requires a relatively small capital investment, but frequently suffers from low extraction ratios. In room-and-pillar mining, coal has been left underground for such reasons as ground control, extraction difficulties, prevention of surface subsidence, and changes in economic or market conditions. Historically, when all U.S. room-and-pillar coal mines are considered, the average extraction is less than 55% (Bieniawski, 1982). This figure may decrease with time as reserves at shallow depths are worked out and mining moves deeper, because room-and-pillar methods inherently require larger pillars with depth to combat the increasing cover load (Figure 1). If room-and-pillar is to continue to be a valuable and viable mining method the declining extraction during development must be offset when possible by partial or complete pillar extraction.

In mining method selection, the alternatives to room-and-pillar mining are either longwall or shortwall methods, which accomplish a nearly total extraction but are



Depth of Mining versus Percentage Extraction of Coal from Development Only Operation.

Figure 1.1

capital intensive and relatively inflexible. Also, longwall and shortwall methods are impractical under some roof conditions, and invariably result in maximum surface subsidence. Room-and-pillar methods have the potential for causing anywhere from zero to maximum surface subsidence depending on the amount of coal extracted. The selection of the appropriate method ultimately must be based on economics. This involves, among other things, a detailed evaluation of the impact of ground control on the mining system, and vice versa.

The challenge to man in the coal mining industry lies in achieving the increased underground extraction goal at a minimum cost. The industry's cause would be lost if it assumed that pillar extraction or higher extraction rates inevitably are associated with higher costs. The fundamental problem in pillaring is that unsupported roof area increases during pillar extraction; therefore, ground control costs also increase. Nevertheless, pillar extraction when practical can result in decreased operating costs, increased utilization of reserves, and an extended life of the mine. Pillar extraction is highly productive because supply, haulage, ventilation, and power systems are already established, and another advantage is the knowledge gained during panel development in such areas as roof and ground behavior and hydrologic factors (Thompson, 1984).

To fully evaluate the potential for pillaring under a variety of ground conditions, a cost model or computer simulator available for pillar extraction cost estimation in room-and-pillar coal mining is needed. The model (PILCOST) presented in this dissertation has the capability of estimating pillar mining costs for new mine properties or the extraction of existing pillars, and is an essential step in the process of mining method selection and evaluation. The model may also be used in selecting the optimum pillar size and degree of surface damage to minimize cost of extraction.

The research work will provide the necessary and sufficient knowledge for the mining engineer today to:

--- ensure optimal utilization of the available resource,

--- produce coal safely,

--- produce coal at an acceptable cost,

--- ensure security of supply from the mine he designs,

--- maintain the necessary flexibility of production,

--- minimize damage to valuable surface.

2. LITERATURE REVIEW

Information related to retreat mining or pillar extraction is limited. Several studies have been made by various investigators during the last four decades. Keenan (1949) has described the successful extraction of pillars at the Kenilworth coal mine in Utah. As one of the early investigations, Keenan's study covers topics such as mine conditions, mine layout, pillar extraction (including the advantages of a 45-degree line), roof support, machinery, haulage, and ventilation, as well as the problem of spontaneous combustion of coal near the surface. Gilley and Thomas (1951) studied the possibility of pillar extraction with roof bolts and described the pillar recovery practices at a mine in southern West Virginia. Tables that compare the results of mining with conventional timbering to mining with roof bolting were also included in the report. Haley, Shields, Toenges and Turnbull (1952) have discussed recovery in coal mining and roof action and control. They have also presented descriptions of mines and the mining and pillar extraction methods used by each.

The first pillar extraction with continuous machines was discussed by Youkins (1952), who described the pillar extraction method employed with continuous machines at a

Pennsylvania mine. Hess (1955) studied the pillar extraction in the Pittsburgh seam with continuous miners and describes the advantages of using the continuous miner instead of conventional mining in pillar extraction. Jones (1955) also studied the pillar extraction method, used at the Lancashire No. 15 mine in Pennsylvania, and has discussed his investigations which include timbering, ventilation, safety, production, and continuous miners. Buckhorn coal mine in Illinois was selected for pillar extraction studies, and a description of the mine conditions, pillar recovery method, the roof support, and roof coating systems has been presented. A productivity study has been done at the Isabella mine, Pennsylvania, where open-end pillaring with a continuous miner was the common recovery method used. The report of the investigation describes various aspects and methods of pillaring with continuous miners, such as stepped pillar line, combination advance and retreat, full retreat, angle plans, conventional open ending, angle open ending, splitting plans, pocket and wing plans, and a basic room-and-pillar plan modified for an extensible-belt. Valeri (1957) also studied the retreat mining system, used at Nemacolin, Pennsylvania, and has reported the investigations in his paper. Mine conditions, roof control, and ventilation are among the topics covered. The system involves the use of conventional units and continuous miners and an open-end pillaring method. Hoard and Cressman (1959) and Norris (1960) investigated an increase in production at Mine No. 41 of the Bethlehem Mines Corporation in West Virginia due to the use of a boring-type miner for pillaring, adoption of slabbing as the pillar-extraction method, a change from a 45-degree to a flat pillar line, and the adoption of a rope-frame panel belt to cut delay time to the absolute minimum.

Stahl (1960) discussed various pillar extraction methods used in Pennsylvania with continuous mining machines. Both common practices and safe practices are discussed in his paper. Laird (1962, 1963) described the pillar extraction method used at the Federal No. 1 mine in West Virginia. The paper describes how continuous miners that are more powerful and rugged boost productivity. Reeves (1966) studied the continuous mining of pitching coal seams and has described the mine conditions, mining method, mining sequence, pillar recovery and ventilation at the Dutch Creek Mine in Colorado. The pocket-and-wing pillar extraction method used to mine the High Split seam in Kentucky has been described by Schroder (1966),who explains the advantages of the system and stresses major considerations during extraction. The pocket-and-fender method of pillar extraction with continuous miners used at the Mathies mine in Pennsylvania is described by Bobo (1966).

Curth (1967) conducted tests at the Nemacolin mine in Pennsylvania to determine the relative pressure changes in the coal pillars during extraction and has presented the conclusions in his paper. A description of the pocket-and-fender method of pillar extraction of a coal seam varying in thickness from 10 to 14 feet is given by Flint and Taylor (1971).

Adler and Gallimore (1972) mentioned the need for a new mining system. Their article provides a general description of both the development and retreating phases of room-and-pillar mining. A cut sequence for wing-and-pocket pillaring, and a pillaring sequence for mining rooms and a single row of blocks with a continuous miner are illustrated. However, because of the disadvantages of the room-and-pillar system, a retreat mining system combining the longwall and room-and-pillar methods is described and recommended.

Stephenson, Gregory, and Riva (1972), Cassidy (1973), and Davis Associates (1975) compared and evaluated the safety aspects of the open-end and split-and-fender systems of pillar extraction. Basic data required to decide what constitutes an adequate pillaring plan that will optimize efficiency, recovery, and safety have been established.

Recommendations have been made to improve existing pillaring methods and to ensure that proper techniques are implemented (Kauffman, Hawkins, and Thompson,1981).

Salamon and Oravecz (1976) and Fauconnier and Kresten (1982) described pillar extraction methods used in South Africa. Recently, the United States Bureau of Mines has prepared a room-and-pillar retreat mining manual for the coal industry (Kauffman, Hawkins, and Thompson, 1981).

Several studies investigate coal production costs in development sections; however, there are very few that provide pillar extraction or retreat mining production costs. Green and Palowitch (1977) made an economic comparison between room-and-pillar mining pillar extraction and shortwall mining. Over the past 15 years, The United States Bureau of Mines has investigated underground mining coal production costs and has published a series of model mine cost estimates. Each study utilized an engineering estimate approach to evaluate costs. These studies assist in identifying coal mining cost components based on specified operating characteristics (Haskins, 1978).

The following discussion reviews the choice of underground mining methods, mine planning and retreat mining, and rock mechanics of total extraction.

2.1. THE CHOICE OF UNDERGROUND MINING METHODS

Because the more shallow coal resources are being depleted, companies must develop deeper deposits and increase percentage extraction to maintain production levels. Selection of mining method and optimizing productivity for mines at depth presents special problems. In room-and-pillar mining, increased pillar size may be countered by increasing the room width to maintain or improve the percentage of underground coal extracted. However, these actions can cause ground control problems that frequently result in serious interruptions in production. Mining costs resulting from such ground control practices increase with the higher extraction ratios for mines at depth, indicating there is an economic optimum above which higher coal revenues are more than offset by the higher mining costs. For design purposes the cost of losses due to high risk ground control must be weighed against the cost of minimizing these problems and improving ground conditions or selecting another method such as longwall or shortwall.

2.1.1. UNDERGROUND MINING SYSTEMS

There are several underground mining methods, but they can be classified into three general groups:

1. Roof supporting methods:
   - -- room and pillar mining
   - -- artificial support or stowing methods
2. Caving methods:
   - -- pillar extraction or retreat mining methods
   - -- longwall/shortwall methods
   - -- sublevel caving methods
3. Yielding pillar methods.

One of the methods listed above is chosen to mine coal reserves by underground methods. The many factors that influence this choice are listed below.

2.1.2. FACTORS INFLUENCING THE CHOICE OF U/G MINING METHODS

1. Geometrical factors
   - -- thickness of overburden
   - -- multiple seams
   - -- seam thickness
2. Geological factors
   - -- primary geological structure
   - -- secondary geological structure
   - -- strata composition above the coal seam
   - -- in-seam partings
   - -- vertical and lateral quality variations

-- variations in seam thickness

-- floor conditions

-- water bearing strata

3. Surface protection

4. Technology

5. Market considerations

-- price of coal

-- quality requirements

-- size grading

6. Size of reserve

7. Capital

8. Labor

9. Availability of equipment

The most common underground mining methods used in the United States are room-and-pillar mining with partial extraction and total extraction methods, and longwall or shortwall mining. Because the objective is to develop deeper deposits and increase percentage extraction to maintain production levels, the area of interest will be limited to caving methods only. A choice will be made from one of the caving methods such as pillar extraction, longwall, and shortwall mining. A comparison of the methods has been presented in Table 2.1.

TABLE 2.1

Comparison of Total Extraction Methods.

SELECTION OF TOTAL EXTRACTION METHOD

Room - and - Pillar with Pillar Extraction	Longwall	Shortwall
-- Small Capital Investment	-- Capital Intensive	-- Capital Intensive
-- Flexible	-- Inflexible	-- Inflexible
-- Potential for Causing Zero to Maximum Surface Subsidence	-- Maximum Surface Subsidence	-- Maximum Surface Subsidence
-- Partial to Complete Pillar Extraction Depending on Roof Conditions	-- Impractical Under Same Roof Conditions	-- Roof Strong Enough to Support Wide Exposed Area
	-- High Operating Cost	-- High Operating Cost
-- Decreased Operating Costs	-- Highly Mechanized	-- Medium Mechanized
-- Seams No Less than 4' Thick	-- Narrow Seams Only	-- Medium Thickness of Seam
-- Gradient Limits	-- Excellent Ventilation	-- Excellent Ventilation
-- Nongassy Seams Only	-- Increased Safety	-- Less % Extraction than Room-and-Pillar Mining with Pillar-Extraction
-- Longer Training Period	-- Short Training Period	
-- Controlled Interaction Between Adjacent Seams		-- Soft Coal Only with Hard Floor
		-- Increased Safety
		-- Short Training Period

2.1.3. ECONOMIC COMPARISON OF DIFFERENT MINING METHODS

In selecting a mining method, economic factors are the most important. A mine can not exist in a free-enterprise economic system, operating under the laws of supply and demand, unless the value of coal produced covers the cost of mining and other related expenses. The ultimate aim in a coal mining operation is to maximize profit. A low-cost-per-ton mining method will be the best choice.

Very few literature studies are available comparing the cost of mining coal by the three different mining systems: shortwall, room-and-pillar with retreat mining, and longwall.

Green and Palowitch (1977) compared the shortwall method to the room-and-pillar method with total extraction on a cost basis at the Hendrix No. 22 Mine of the Beth-Elkhorn Corporation. The study consisted of a four-year program from 1973 to 1977, which developed and extracted six panels in a seam 48- to 54- inches thick which lay under 150 to 800 feet of cover. The main objectives of the study were to prove the system and compare costs, productivity, safety, and extraction rate with existing methods.

Shortwall mining was demonstrated to be a viable alternative to room-and-pillar mining under the same mining conditions, working a double shift daily. The profitability of the shortwall mining system was found to depend not only on the increased depreciation cost of the powered roof supports and the decreased cost of supplies and materials, but also on such factors as the fixed and variable indirect costs and the selling price of coal.

The shortwall production averaged 975 tons of raw coal compared with 905 tons from a room-and-pillar unit under essentially identical conditions. The estimated direct operating costs per ton of raw coal were \$4.03 and \$4.21 for room-and-pillar and shortwall methods, respectively. Since the rate of recovery of shortwall is higher than that of room-and-pillar method, Green and Palowitch concluded that the total cost of shortwall will be lower than room-and-pillar with pillar extraction. The report indicates an extraction rate of 82 percent for the shortwall method and about 85 percent when using pillar extraction as a second phase after having used room-and-pillar method as the first phase.

The shortwall method was successfully applied at the Homer City Mine of Helen Mining Company in a seam 50- to 54- inches thick at a depth of 650 to 820 feet. Although no cost figures are available, management concluded that the shortwall method was highly competitive in comparison with alternative methods used on the same mine under essentially identical conditions.

Fauconnier and Kersten (1982) compared the shortwall method with the bord-and-pillar method using pillar extraction on a cost basis under South African economic and mining conditions. The bord-and-pillar data were obtained over a long period with both development and pillar extraction practiced simultaneously.

In that study, they found that the capital requirement for shortwall equipment for every ton mined was higher by a factor of 1.44, when compared with similar production by a bord-and-pillar unit. The ratio of men involved in shortwall to bord-and-pillar was 1:1.76 for similar production. The cost-per-ton mined was of the ratio of 1:0.936. Because the labor cost was very cheap in South Africa, the shortwall method did not seem to be a viable alternative to the bord-and-pillar method.

An economic comparison of longwall mining with pillar mining and shortwall mining methods is complicated and is based on a number of assumptions regarding panel development, depreciation of capital, geographic expansion, escalation of different cost components, etc., which might vary for different applications (Fauconnier and Kersten, 1982). Depth dictates the mining method in most of the countries of Europe, Asia, and Africa because of low percentage extraction from room-and-pillar workings. A definite extraction break-even line can be drawn between longwalling and room-and-pillar mining. The effect of longwall equipment is also important to the extraction break-even line. If the reach of longwall equipment is up to the full seam thickness, the percentage volumetric extraction will be higher.

Uasuo Tsuruoka and Masamiti Shikasho (1980) developed a formula to optimize the face length of longwalls and classified the costs associated with longwall mining in the following categories:

(i) costs $\propto$ face length (L)

these costs are depreciation and maintenance costs of face supports and face conveyor,

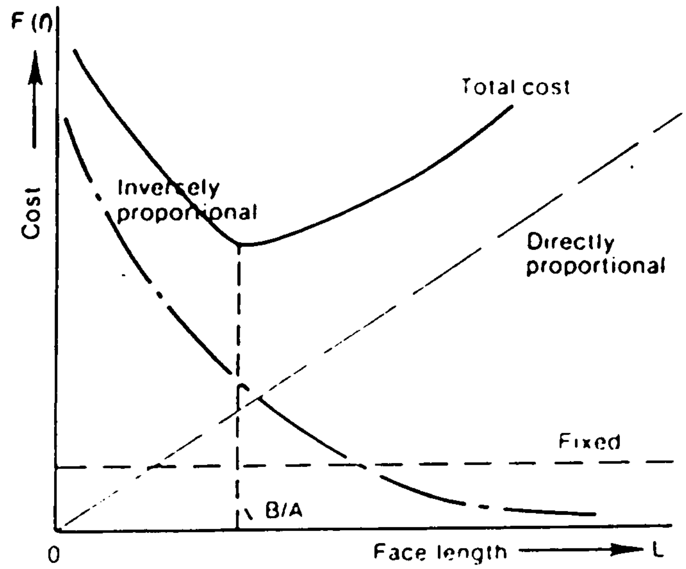
(ii) costs $\propto$ 1 / facelength (L)

these costs are depreciation and maintenance costs of the face cutting machine, gate road conveyor development costs, and power supply costs for face equipment,

(iii) fixed cost: cost of longwall face move.

Total cost at a face, $f(L) = AL + B/L + C$

where, A, B, and C are proportional constants for (i), (ii) and (iii). This total cost equation is shown by the solid line in Figure 2.1. The optimum value of face length L, can be obtained by differentiating the total cost



Optimising longwall face length in terms of cost. (After Fauconnier, 1982)

Figure 2.1

equation and equating to 0.

$$f'(L) = \frac{d}{dL}(AL + B/L + C) = 0$$

$$= A - B/L^2 = 0$$

Therefore, $L = \sqrt{B/A}$.

The initial capital cost per annual ton of longwall mining is 167 percent of room-and-pillar mining with pillar extraction. The direct operating cost (or in-section cost) of longwall mining is 120 percent of pillar extraction (Kersten, 1982).

Sathaye and Vyas (1983) compared the productivities of bord-and-pillar section and mechanised pillar extraction with load-haul-dumpers section. The ratio of daily productions from a pillar extraction section to that from a development section was 1.25. The cost of pillar extraction per ton of coal was lower than the cost of development for daily productions greater than 330 tons.

Flint, Buchan, Strachan, and Clarkson (1984) proposed a technique of partial pillar extraction with controlled goafing of the superincumbent strata. Geological conditions in some of the mining areas are such that conventional pillar extraction cannot be practiced safely or efficiently. As an alternative to total pillar extraction, a system of partial pillar extraction using mechanized-mining methods with modified goaf-line support has been evolved. The coal pillars remaining after partial extraction of the pillar, together with the installation of a double line of full-column resin roofbolts, ensure control of the edge of the area that has been mined out. Where roof conditions preclude the use of conventional pillar extraction methods, this method holds promise as a safe, efficient, and economically feasible alternative for the recovery of coal from pillars and for an increase in volumetric extraction of the reserves within a mining panel.

Trueman (1984) compared the productivities and costs of different mining methods for thin coal seams. The ratio of pillars developed to pillars extracted during a given time is the important factor which determines the attractiveness of pillar extraction process. If the ratio of pillar extraction to pillar development productivities is less than one, the in-panel cost per extracted ton of coal is cheaper for development-only method.

Smith (1985) studied the mechanised pillar extraction at The Durban Navigation Collieries (Pty) Ltd. and found that the ratio of pillars developed to pillars extracted was 1.0 pillar developed for every 2.7 extracted. Mechanised pillar extraction was replacing the handgot methods previously used at the Durnacol Colliery. Improvements had been achieved in production, productivity, safety, cost-per-ton, and percentage extraction of reserves.

2.2. MINE PLANNING AND RETREAT MINING

Room-and-pillar retreat mining has been practiced successfully in the United States and abroad for years. The retreat mining techniques vary widely over the United States. The techniques used in a particular area are dependent upon specific local conditions, the influence of experiences of adjacent mines, and local Mine Safety and Health Administration roof control specialists.

The objective of the pillaring operation is the complete extraction of the coal seam, having due regard for profit and safety. There are several factors which affect the decision to plan for retreat mining. Among these factors, the following are important (Kauffman, Hawkins, and Thompson, 1981):

- geology
- environment
- labor
- economics

These factors are considered in detail while making the decision to retreat mine.

2.2.1 GEOLOGIC CONSIDERATIONS

The nature of the coal and the rocks in which it is imbedded affect the decision to retreat mine. The different sediments of rock found in a given formation of coal deposit, the composition of rock sediments above coal, the rock layer under the coal (mine floor), and multiple coal seams in the same formation must be considered (Kauffman, and Hawkins, 1981). Some common rock types associated with coal seams and their properties as they relate to retreat mining are discussed below.

Rock Strata:

Coal: - A coal seam may consist of different thin layers of coal which have varying physical properties and hence create high internal stresses and cause spalling of exposed surfaces when mining. If the coal contains a high percentage of ash, it is strong, dense and dull, whereas bright shiny coal is weak. The luster in between bright and dull indicates a strength between the two separate strengths. Different coals have different face cleavages. The number of fracture planes also differs from coal to coal. The strength of coal is important in the design of pillars and also in determining the susceptibility of the coal to bumps or bursts.

Shale: - It forms the immediate roof for most of the coal deposits. If the thickness of the shale roof is more than eight feet it will be a good roof (Hawkins, 1981). This is because a thick layer resists permeation by water. However, if the shale stratum is thin, particularly if it has fractures, moisture can penetrate. Most shale deteriorates due to absorption of moisture when exposed to humid air. The adhesion of the shale stratum to the rest of the overburden is reduced by water. The shale roof is good for retreat mining since a clean, tight fall can generally be attained.

Slate: - It can be strong or weak and may contain irregularities. Slate has a tendency to separate from other layers above it, causing hazardous conditions. Generally it makes a good roof for retreat mining since a clean fall can normally be attained. Extra support measures, however, are required during advance mining when a fall is undesirable.

Sandstone: - If the sandstone grains are well cemented, it makes a good roof during advance mining and a poor roof during retreat mining. If the grains are poorly cemented then it makes a good roof during retreat mining and a poor roof during advance mining. Additional problems can be encountered when the sandstone contains water. The presence of water can cause the sandstone to lose its adhesion to the rest of the overburden, resulting in a hazardous roof condition. When water-bearing sandstone overlies gray shale, the water may weaken the shale below and cause severe roof instability.

Bone Coal: - This is a mixture of shale and coal usually found at the top of a coalbed. If bone coal is found overlying the coal, it is used for the roof. The strength of the bone coal is dependent on shale content. If shale overlies the bone coal then it is good for retreat mining.

Limestone: - This is less disturbed by faults than other formations. Limestone does not often form the immediate roof above the coal to be mined. It often forms the main roof and, like sandstone, may be a poor roof for retreat mining because of its massive nature and the resulting difficulty in getting clean falls.

Soapstone: - This consists primarily of impure talc and can be white, gray or greenish. Soapstone is a very weak rock because in a roof it cannot support its own weight. It can become a source of many hazards if present as either an immediate or main roof.

Fire Clay: - Fire Clay is formed from a deposit of mud under the effect of time and pressure. Its color varies from white to deep gray. It may occur over the coalbed or form the floor. For most coal seams, it forms the immediate floor. It can create hazards when present in the roof because of its poor adhesion to other strata and lack of strength.

Overburden Composition:

The overburden affects pillar dimensions, equipment selection, roof control plan, and the selection of mining technique. The overburden should be strong enough, given the proper pillar sizes and temporary support, to allow the development work to be done safely, yet weak enough to cave at the proper time during retreat mining. The overburden composition affects safety since a very weak overburden is dangerous and could fall prematurely during the pillar extraction process. A very strong overburden that does not cave soon after extraction of a pillar, such as massive sandstone, fails to relieve the pressure upon remaining pillars. This could cause the remaining pillars to fail, resulting in a dangerous, unplanned, and massive roof fall, resulting in the loss of much coal. Overburden composition also affects the economics of retreat mining in that either the cost of supporting a very weak immediate roof might be prohibitive, or the cost of caving a very strong overburden might be too high. A list of overburden compositions and roof qualities for various coal fields has been presented in Table 2.2.

Floor Composition:

Often the floor consists of a fire clay which in the presence of water might become mud during retreat mining

TABLE 2.2

Overburden Composition & Percentage Extraction

Location	Name	Overburden Composition	Immediate Roof	Thickness	Percentage Extraction
McDowell County W. Va.	Pochahon. No. 4	Shale 5' & Sandstone 30'	Slate	29"	80%
Raleigh County W. Va.	Beckley	Shale & Sandstone	Shale	50"	85%
Raleigh County W. Va.	Beckley	Shale 100' & Sandstone 50'	Shale & Slate	100' 2"	87-90%
Dicken. County Va.	Upper Banner	Sandstone	Sandstone		50%
Kanawha County W. Va.	No. 2 Gas	Shale & Sandstone 28'	Sandstone		50%
Kanawha County W. Va.	Winifrede	Shales & Sandstone	Shale Sandstone	0-14" 75'	65%
Kanawha County W. Va.	Winifrede	Shale & Sandstone 75'	Shale Sandstone	0-35" 75'	60%
Kanawha County W. Va.	Coalburg	Shale & Sandstone	Shale	6-8'	75%
Wise County Va.	Taggart	Shale & Massive-Sandstone	Shale	3'	75%
Harlan County Ky.	"C" Taggart	Sandy Shale and Sandstone	Sandy-Shale	12'	80%
Floyd County Ky.	Elkhorn No. 2	Sandy Shale and Sandstone	Sandy-Shale		80%

- (continued) -

Cambria County Pa.	Upper Kittan-ning	Shales & Sandstone	Sandy-Shale		85%
Indiana County Pa.	Upper Freeport	Shales & Sandstone	Dark Shale	10'	80%
Alabama	Thompson	Slate, Shales & Sandstone	Slate	4.5'	60%
Monong-lia County W. Va.	Sewickly	Shaley Draw-slate, Shale, Thin coal, Limestones & Sandstones	Shaley-Drawslate to Sandstone	5.5'	85%
Mingo County W. Va.	Upper Cedar	Shale 35' & Sandstone 35'	Drawslate	0-3'	85%
Harlan County Ky.	"D" Seam	Shales & Sandstone	Grey Sandy Shale		75%
Harlan County Ky.	"B" Seam	Shales & Sandstone	Sandy-Shale to Sandstone		90%
Lee County Va.	NO. 10 Seam	Sandy Shale and Sandstone	Drawslate Coal Sandstone & Slate	5" 10"	75%
Pike County Ky.	Pond Creek Seam	Shales & Sandstone	Firm Slate		90%
Alabama	Gholson	Shales & Massive-Sandstone	Very Firm Shale	3'	85%
Alabama	America	Shale & Sandstone	Very Weak Shale	8-10'	60%
Greene County Pa.	Sewickly	Shale, Limestone & Sandstone	Very Firm Slate	5'	90%

and thus impede the movement of mining machinery. The fire clay may also exhibit plastic flow under excessive pillar stresses, causing the pillars to sink and the floor to heave. Ideal mining conditions exist in mines with firm shale floors and sufficient yield in the roof, pillars, and floor. But if yield is low and overburden pressure is high, extremely hazardous conditions can result.

<u>Pitching Seams</u>:

The pitch of a seam can be a factor in determining whether to retreat mine. Steeper grades more than seven percent increase haulage effort and time and decrease the amount of material that can be moved. The seam pitch also affects the pillar shape and weight distribution.

<u>Multiple Seams</u>:

Multiple seams of coal can have a significant influence on the decision not to retreat mine. If mining of multiple seams is planned in a virgin coalfield, the proper order of mining is from top to bottom with total extraction if possible. It is also advisable to mine one seam at a time. Prior mining of seams might make retreat mining undesirable and perhaps even impossible.

<u>Geological Conditions of Different Coalfields</u>: - The most important geological conditions affecting the extraction of coal are the following (Fauconnier and Kersten, 1982):

- depth of Coal,

- thickness of coal seams - too thin, too thick or variable,
- parting thickness between coal seams,
- geological structure: primary structure - original-floor
  secondary structure - dykes, sills, faults,
- in-seam partings and split seams,
- selected quality horizons,
- floor rolls and roof rolls,
- wash ins and wash outs,
- roof conditions and floor conditions.

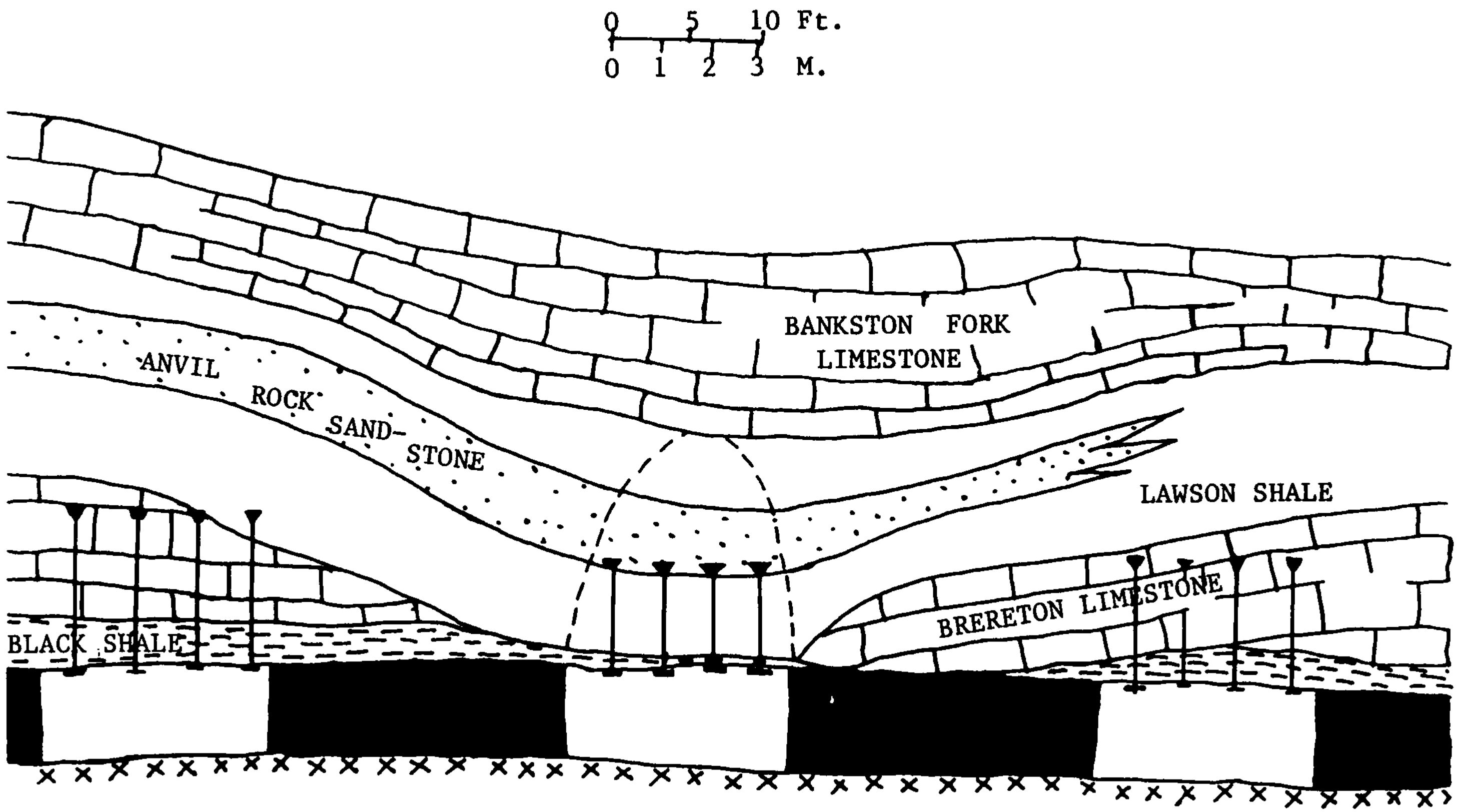
The characteristics and qualities of the coal seams generally vary from one coalfield to another due to differences in depositional environments in the individual coalfields. The geological conditions for major Appalachian coalfields of the United States are discussed below.

Illinois Coal Basin

All the extensive underground mining in the Illinois Coal Basin has been confined to a few coals occurring in the Carbondale Formation which is found near the middle of Pennsylvanian System. The most important coals are the Herrin (No. 6) Coal (No. 11 in western Kentucky and the Herrin Coal in Indiana) and the Springfield-Harrisburg (No. 5) Coal (No. 9 in western Kentucky and Springfield Coal V in Indiana). The most important characteristics of the coal from these coalfields are persistence of thick, relatively parting free coals; gentle dips; relatively dry conditions; low methane liberation; and generally good roof conditions. The relatively soft floor (poorly consolidated underclay) is an undesirable characteristic (Hopkins, 1980).

The thickness of the coal presently being mined varies from 4 feet to 8 feet, with the average being 6 feet. Most of the present underground mining is in areas where the No. 6 Coal is 300 to 500 feet deep but several large mines are in the 600 to 800 feet range. Over most of the coal basin, the coals are overlain immediately and abruptly by up to 5 feet of hard, black, "slaty" shale which is, in turn, overlain by a limestone unit of similar thickness in the case of No. 6 Coal and generally less than 2 feet in the case of the No. 5. These roof rocks contain marine fossils. At many places in No. 6 Coal, the black shale pinches out and the limestone constitutes the immediate roof. The other roof type is silty gray shale up to 60 feet thick. This shale occurs between the coal and black shale/limestone.

<u>Black Shale/Limestone Areas</u>: - The No. 6 Coal in Illinois has stratigraphic variation in the immediate roof. Local depositional conditions resulted in a lenticular occurrence of the Brereton Limestone as depicted in Figure 2.2. The



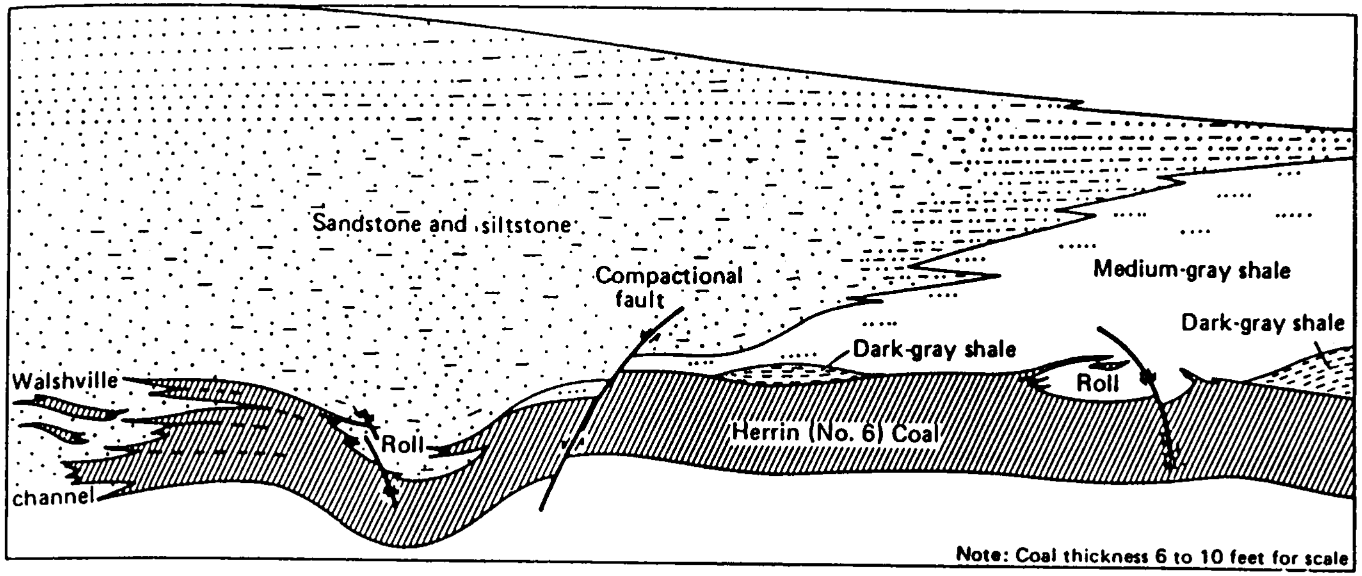
Generalized section of strata over No. 6 coal in area of Black Shale/Limestone Roof

Figure 2.2

Brereton Limestone has pinched out and the Black Shale has thinned. The Lawson Shale, which is weaker and has abundant slickensides, has much more influence on the roof quality over the central mine entry (Hopkins, 1980). Normal conditions, where the limestone and black shale are both present, are found in the two outside entries, and roof quality should be good. There are many areas in the coal seam where Brereton Limestone is the immediate roof and roof conditions are excellent. At some places, Anvil Rock Sandstone has cut down through the Brereton Limestone and lies very close to the coal. Problems arising in these areas are (i) water leaking down through the boltholes (ii) a potentially weak roof (iii) fractures generated by differential compaction in the roof.

Gray Shale Areas: - The gray shale roof type includes several distinct lithologies (Figure 2.3): (i) lenses of dark gray, thinly laminated shale; (ii) large areas of medium gray, poorly laminated shale, silty to nonsilty; and (iii) areas of sandstone and siltstone, often planar-bedded. Other rock types have also been observed in mine roofs, but they are less common than those mentioned above.

Additional problems appear when the sandstone contains water. Where water-bearing sandstone overlies gray shale, the water may weaken the shale below and cause severe roof



Section of lithologies and structures of gray shale roof type,(After Damberger)

Figure 2.3

instability. Other features hampering mining near the Walshville channel include split coal, steeply dipping coal, large rolls, wash outs, and compactional faults (Damberger, Nelson, 1980).

Southern West Virginia and Eastern Kentucky Coalfields

According to Horne, Firm, and Gruccio (1979), the best quality roof conditions in this region of the Appalachians occur in hard graywacke sandstones that are more than 10 feet thick and extend horizontally more than 2000 feet. These sandstones were deposited in active, laterally migrating channels. Lag deposits composed of shale and coal pebbles commonly formed near the base of the channels. These lags can weaken the sandstone and cause roof problems during advance mining but easy roof conditions during retreat mining. Unjointed, well cemented, orthoquartzitic sandstones with similar thickness and areal extent as the graywacke sandstones, also may provide excellent roof conditions during advance mining. Unfortunately, they usually are jointed and fractured, and in this state, the resulting blocks come loose causing severe roof falls.

In flat bedded sandstone and interbedded sandstones and shales, the roof quality is dependent on bed thickness. If the beds are less than 2 feet thick, parting separations can occur along bedding planes, making bolting necessary. Where the beds are 2 to 10 feet thick, the roof conditions are excellent because bridging strengths are sufficient to prevent falls. However, where bed thicknesses exceed 10 feet, slickensided surfaces may develop due to differential compaction, and failure may occur along these surfaces. Coarsening-upward rock sequences that grade from shale upward through shales with thin sandstones streaks to interbedded sandstone and shale, capped by sandstones, provide few roof support problems. However, separation at sandstone shale bedding planes can produce roof falls. Hence, roof bolting is an essential precaution during advance mining.

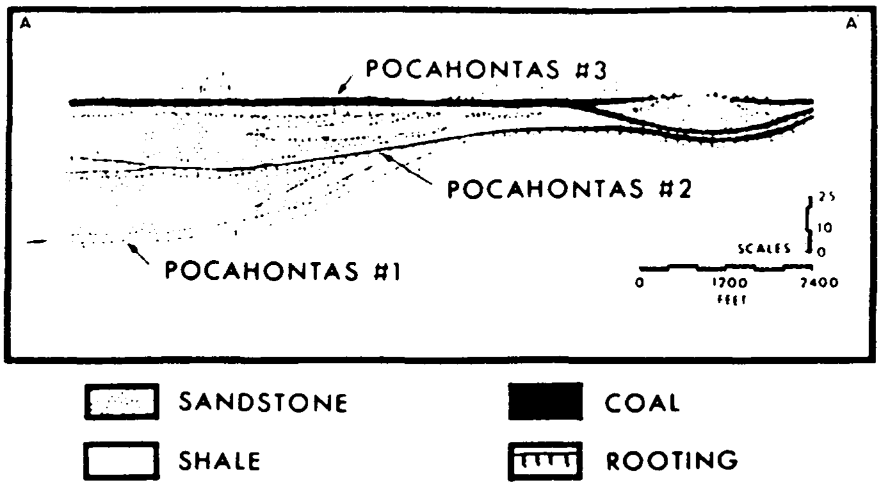
In some places, the coals are overlain by a brittle nonbedded, carbonaceous black clay stone that is jointed. Blocks of this rock may suddenly come loose causing dangerous falls. Another roof problem occurs during advance mining when fine-grained rocks such as shales, siltstones, and shales with sandstone streaks are extensively formed along the bedding plane in a near-cylinderical shape called burrows. The burrow structures can reduce significantly the strength of these fine-grained rocks and cause rooffalls.

In areas where less compactible coarse-grained sandstones are present as discrete bodies in more compactible fine-grained sediments, slickensided surfaces form at the contact between the two rocks. Zones of weakness are developed along these surfaces, and separations may cause severe roof falls. Another place where severe roof problems may develop is where channel-bank slump blocks form the roof over the coal (Horne and Ferm, 1979). Because of the numerous slickensided surfaces and the size of the blocks, severe roof problems can be expected. Roof bolting is of little use under these conditions.

<u>Pocahontas #2 and #3 Coal Seams, in Southern West Virginia and Southwestern Virginia</u>

The Pocahontas #2 and #3 seams comprise a couplet that makes up the most widespread coal body in the Pocahontas Basin. The study by Ferm, Staub, Baganz, Clark (1979) encompasses the coalfields of this area. The Pocahontas #2 is separated from the overlying Pocahontas #3 by rocks about 25 feet thick, but in the northern part the two seams are only 2 feet apart and at some places they merge into a single seam. A typical cross-section of the seam is shown in Figure 2.4. In the northwest direction, the seam thins as it passes into dark shales, and then into red and green shales. Pocahontas #3 is generally thicker and more widely distributed than the Pocahontas #2.

The kinds of rock types found above the Pocahontas #3 vary from hard, brittle othoquartzite sandstone and greywacke sandstones to fine-grained shale and seatearth. Grey sandstone in thicknesses greater than 15 feet, which



Cross-section of the Pocahontas #2 and #3 seams (After Ferm).

Figure 2.4

is within bolting distance of the coal, makes good top (Melton, 1979). Sandy shale is a rock type which makes excellent top when found in a rapidly coarsening upward sequence. However, where sequences of sandy shale exceed 25 feet in thickness, they are susceptible to jointing and break easily. Fine grained rocks, shales, and fire clays are not abundant roof rocks in Pocahontas #3 mines. Rashy shale is common as thin deposits directly above #3 seam. It is common to mine the "rash" along with the coal because it normally spalls and flakes off causing a debris problem. Sandstone-filled channel scours have been found to have an adverse effect on the roof. They are linear trough-shaped features that usually cut into shales above the coal. They are called "rolls" or "wants". Some of the worst roof conditions in the Pocahontas #3 mines were produced by slickensides.

The Beckley Seam in Southern West Virginia

The thickness of the coal seam varies from 3 to 18 feet within a distance of 150 feet in some mines. The seam is split into different benches at some of the mines. Some partings between the benches of a split coal seam are continuous, and thickening of the partings is accompanied by a corresponding thinning of coal. The rock above the coal seam is mainly graywacke sandstone 40 to 100 feet thick. At some places in the area graywacke sandstone cuts down into the coal and a similar pattern is followed by orthoquartzite rock (Ferm et al, 1979).

The Sewell Seam near Beckley, West Virginia

This seam lies about 250 feet above the Beckley seam, and is generally uniform in thickness. Rocks lying beneath the Sewell are mainly a coarsening upward sequence about 60 feet thick, punctuated by an occasional linear body of graywacke sandstone (Galloway, 1972; and Jones,1975). The seam thins and splits into a shale-dominated sequence towards the northwest direction.

The Allegheny Seam in Western Pennsylvania

The body of the coal seam is about twenty four miles wide and more than 5 feet thick. The thickness varies to 3 feet at some places within the area. Rocks lying beneath the seam are limestones or sandstones which lie within fifteen feet of the bottom of the seam. At the margin of the deposit, these rocks are replaced by shale and some thin sandstones.

Rocks overlying the seam are dark plant-bearing shale directly overlying the coal coarsening upwards into siltstones and sandstones. In some places sandstones cut downward through the dark shales and siltstones and into the coals.

2.2.2 ENVIRONMENTAL CONSIDERATIONS

Retreat mining resulting in increased underground extraction of coal frequently has an adverse effect on the environment. These effects may be due to problems in mine waste disposal, acid mine drainage, subsidence and underground hydrology. Two factors of primary importance in retreat mining operations are underground hydrology and subsidence.

Underground Hydrology

Whenever the overlying strata are disrupted due to roof collapse or subsidence, the underground hydrology is affected. Fissures and cracks formed due to caving allow surface run off water to more easily permeate the overburden and create pollution-causing acidic effluents (Kauffman & Hawkins, 1981). Analysis of the overlying strata prior to retreat mining can help to avoid acid-forming situations. Rock fracture zones and faults have a strong influence on ground water flow patterns, and often the fractures and faults collect and convey large quantities of water. Avoiding these areas by leaving pillars in place may alleviate serious problems.

A complete hydrogeologic site evaluation to determine aquifer characteristics and waterflow systems is necessary before making the decision to retreat mine because most underground mines receive water from overlying aquifers. Retreat mining causes roof collapse and severely disrupts the natural aquifer conditions.

Aquifer Characteristics: -

An aquifer is an underground formation of specific dimensions which contains ground water that can be extracted under the infuence of gravity.

There are four basic parameters attached to an aquifer: Hydraulic Conductivity, Storage Coefficient, Transmissivity and Drainable Porosity (Rakesh, 1980).

1) Hydraulic Conductivity or Permeability: the amount of flow per unit of cross-sectional area under the influence of a unit gradient.

2) Storage Coefficient: the volume of water released or stored per unit of surface area of the aquifer per unit change in the water level.

3) Transmissivity: the average permeability times the thickness of the aquifer.

4) Drainable Porosity: this parameter indicates how much water is available if a section of the unit desaturates.

Dewatering of an Aquifer - Theim Formula (Rakesh & Lele, 1980):

Assuming a steady flow, uniform in horizontal and radial directions, isotropic and with no entrance loss entering the well, the rate of inflow can be calculated by:

$$Q = \frac{K\ 2\pi m\ (D_1 - D_2)}{C\ \mu\ \log_{10}\ (R/r)}$$

where: Q - rate of flow (ml/sec)

K - hydraulic conductivity (Darcy)

m - average saturated thickness of the aquifer (cm)

R - radius of effect (m)

r - well radius (m)

C - constant (2.3)

μ - viscosity (centipoise)

D_1 - drawdown at nearer observation well (m)

D_2 - drawdown at farther observation well (m)

T - transmissivity (metre/sec^2.

Groundwater associated with sandstones: - Groundwater in sandstones may be contained either within pores in the sandstone or in cracks intersecting the sandstone. Below the table all the pore space is normally occupied by groundwater. During drainage, 90 to 99 percent of this water is retained within the sandstone because of retention forces (Fauconnier and Kersten, 1982). Only 0.5 percent of rock volume loses water which is drainable. The permeability of the sandstones is very low and hence the rate at which groundwater can be released from the pores is negligible. Yields from boreholes collecting water from the pores in the sandstone are fewer than 0.05 litre per second (Davis & DeWeist, 1970). Cracks in the sandstones are several and the probability of hitting such a crack by a borehole from the surface would be very low. The permeability of cracks in the sandstones is much higher than that of the pores. The geometrical dimensions and frequency of the cracks are important factors determining the yield. If the depth of cracked sandstone bed is very large, the weight of the overlying rocks tend to close the cracks.

Groundwater associated with shales: - The shales are fine-grained and impervious to water and act as confining layers, separating one sandstone from another. The main source of groundwater is via fissures and joints, which are interconnected.

Groundwater associated with limestones: - The limestones are fairly dry and intergranular permeability itself is very low. The main source of the groundwater is via interconnected fissures and joints.

Mining by the Total Extraction Method in the Proximity of Water Bodies:

U.S. Bureau of Mines guidelines (Skelly and Loy, 1976) are applicable to flat seams and seams having an inclination up to 30°. If these guidelines are to be used for steeper seams, the rock strata between coal and waterbodies are tested for strength and cohesiveness. "Rock Quality Designation" can be a useful tool (Rakesh, 1980).

When mining by the total extraction method is to be conducted under the water body, a safe development plan is made and adherence to the following guidelines is required:

"Full extraction should not be attempted unless at least 100 metres of intervening strata exists between the bed and the surface waters or interlying hazardous strata, for each metre of the extracted bed thickness, and the cumulative effect of multiple seam mining does not induce a tensile strain of more than 10 parts per thousand at the floor of the water body."

If mining by the total extraction method is to be conducted under the water body in a shallow deposit coal seam or in a multiple seam area, adherence to these guidelines will result in the prevention of mine inundation:

"Full extraction should not be considered under roof that is shallower than 700 feet unless the overlying strata is to be monitored by techniques such as micro-seismic monitoring. If this technique is used, the overburden can be as shallow as 350 feet, for coal beds that are 1 metre or less in thickness. Any single seam of coal beneath or in the vicinity of any body of surface water may be totally extracted, provided that for each 1 metre thickness of coal seam to be extracted, a minimum of 200 feet of solid strata cover exists between the proposed workings and the bed of the body of surface water. Where multiple seams exist, all may be worked by total extraction provided that for each 1 metre of aggregate coal and rock thickness of all seams to be extracted, a minimum thickness of 200 feet of solid strata cover exists between the proposed workings in the uppermost seam and the bed of the body of surface water."

During pillar recovery in any mine when working places approach within 50 feet of abandoned sealed workings that may contain dangerous accumulations of water or gas, or within 200 feet of any workings of an adjacent mine, boreholes should be drilled to a distance of at least 20 feet in advance of the face of such a working place. Such boreholes should be drilled sufficiently close to each other to ensure that the advancing face will not accidentally hole through into such workings (Coal Mining Act of Illinois, 1971).

Effect of Retreat Mining On Ground Water Resources:

Increased underground extraction of coal often results in a collapse of the overlying strata. The effect of an increase in extraction or room-and-pillar retreat mining on the roofrocks can be explained as a continuous sequence of events (Causens and Garrett, 1969; and Fauconnier and Kresten, 1982):

(i) The shale and sandstone immediately above the coal seam will collapse soon after support has been removed.

(ii) At this stage, cracks will be generated in all the overlying rocks and permeability of the rocks will be increased. Water from the overlying strata will drain into the room and pillar retreat mining area and will be contaminated.

(iii) As the retreat mining face will advance, more cracks will be generated in the roof rocks and the amount of water flow into the mining area will increase.

(iv) Sometimes circular cracks will appear on surface.

(v) The degree of fracturing of roof rocks decreases from bottom to top until single cracks show on the surface.

(vi) The problems created due to influx of water are the following:

- Interference with mining operations
- Boreholes of farmers in surrounding areas depleted
- Water becomes useless for farming due to contamination
- Continuous flow of rain water into abandoned workings.

<u>Chemical Contamination of Underground Water in Areas of Retreat Mining</u>:

In most of the coalfield areas, the quality of water available from boreholes for irrigation is of high standard. This water does not require any treatment before being used for agricultural purposes. As retreat mining advances, the overlying strata fractures and the ground water present in the overlying rocks flows down through fractures and is exposed to fragments of shale, sandstone, limestone, and coal. The sandstone is inactive chemically and has very little effect on the chemical changes. Shale consists of a high percentage of clay, which is negatively charged and absorbs through the ages the positively charged elements such as Na, K, Ca, and Mg (Schuttler, 1977). As the groundwater comes into contact with shale, preferential ion exchange occurs, causing Ca and Mg in the water to be absorbed onto the clay particles in exchange for Na and K, which will go into solution. As the groundwater passes through the fractured shale layers, other anions such as chloride, sulphate, carbonate, and bicarbonate also will go into solution. Hence, the groundwater arriving in mine workings will have a completely different chemistry from the original (Fauconnier, 1982).

The pyrite in the coal oxidizes, resulting in an acid solution, and when acidic water comes into contact with limestone it becomes neutralized. Mine water will not be acid if $CaCO_3$ is in excess supply.

Subsidence

When pillars are extracted in the retreat mining process, the overlying strata caves and fills the void. Depending upon the area affected this may cause surface subsidence in varying degrees. Thus, constraints may be imposed on pillaring by the necessity to protect the surface and structures. Where there are multiple seams, upper seams may be affected. When areas are identified where subsidence is not allowable, such as under roads or buildings, an unmined area or pyramid-shaped zone of protection under the structure is left. This is due to the inverted-cone effect of the subsidence area. If the surface area where subsidence is not allowed is very large, retreat mining might not be practical. Surface subsidence and its effects on the environment have been discussed in detail in section 2.3.

Depth of mining, coal seam thickness and lithology of the overburden are the controlling factors that determine the surface effects of retreat mining. If the mine is very deep, the strain produced on the surface due to mining activities underground will be less; consequently, the disturbance on the surface will be less. The thicker the coal seam, the greater the disturbance on the surface. Subsidence will cause the formation of pans, in areas of flat surface topography and the pans or depressions, will prevent surface runoff of rainwater to nearby streams and rivers. The surface depressions will fill with rainwater during the rainy season, destroying any vegetation that has grown.

2.2.3 LABOR CONSIDERATIONS

Labor is one of the most important issues facing the underground coal mining industry and is one of the primary considerations to be evaluated when making the decision to retreat mine (Kauffman and Hawkins, 1981). There has been a dramatic redistribution in the age of the work force during the last decade. Earlier the over-50 age group accounted for the largest number of workers, while the under-30 age group made up the smallest number of workers. Now, however, this distribution trend has proven to be the cause of problems, such as a loss of mining experience. The older miners who have retired have not trained the new, younger work force in the art of retreat mining operations.

In the process of deciding whether or not to pillar, mine management will have to determine if it has the desire, ability, and patience to gradually and safely develop a retreat mining work force. An experienced retreat mining work force cannot be created quickly or easily; it requires time and patience.

2.2.4 ECONOMICS

A retreat mining section can give a higher productivity than an advance. Most of the equipment required for a retreat mining section must already have been installed during advancing. Only temporary roof supports are required to be installed during retreat mining.

The economic incentive for retreat mining is not only a matter of reduced cost but also of increased recovery. Retreat mining can give recovery as high as 80 percent. The mine engineer must assess the economic impact of retreat mining using an econometric model, preferably a discounted cash flow model (DCF), which considers the decreased value of money over time.

There are several potential advantages that must be evaluated when considering the possibility of retreat mining, which are listed as follows (Hawkins, 1981):

1. Decreased operating costs (productivity rates and supply costs).
2. Increased utilization of reserves (improved recovery rate of coal).
3. Extended life of the mine.

The impact of retreat mining upon these parameters -- operating costs, reserve utilization, and mine life-- can be evaluated using the DCF model. The parameters can be estimated by assigning values to the variables and then each variable may be tested for its sensitivity to changes in other variables attributable to retreat mining techniques. Any economic advantages of retreat mining that are identified will be significant in determining the overall profitability of a coal mining venture.

2.3 ROCK MECHANICS OF TOTAL EXTRACTION

In the room-and-pillar mining method pillars should serve two functions: (a) to support the roof before the pillars are extracted, and (b) to be capable of being extracted safely and efficiently. Development room-and-pillar workings should be designed with a high safety factor (not less than 2) to provide adequate protection to the pillars to be extracted during retreat mining. Until recently pillars in underground mines were designed by experience rather than by engineering methods. Mining engineers learned through years of experience the approximate size of mine pillars that would render the workings safe. To avoid collapse, high factors of safety were often used and therefore unnecessarily large pillars were left, frequently causing loss of coal. Minimizing pillar dimensions has been attempted but has caused pillar failure. These conditions still occur in the United States and many coal mining countries (Sheorey and Singh, 1970).

The South African Chamber of Mines has made considerable progress in the design of pillars (Denkhaus, 1962; Salamon and Munro, 1967; Grobbelaar, 1968; and Bieniawski, 1968). Pillar design presupposes a knowledge of: 1) load applied to the pillars due to overburden strata, and 2) the strength of the pillar material for the size at which the pillar is to exist.

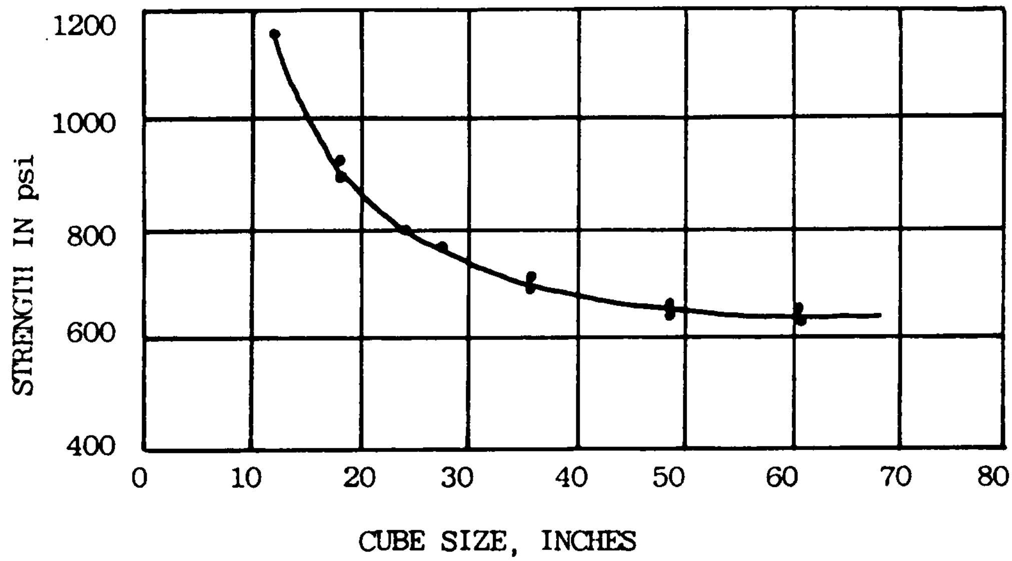
Bieniawski (1967, 1968) has found during the in-situ tests that, for cubic coal specimens, the strength decreases with the increasing size of the specimen while, for a given height of square cross-section specimens, the strength increases with an increase in specimen width. Both these processes tend to an asymptotic strength value; that is, from a certain specimen size onwards there appears to be no more change in the specimen strength in spite of the increasing specimen width. These findings of Bieniawski's in-situ tests have been depicted in Figures 2.5 and 2.6.

2.3.1 ROOF SPAN DETERMINATION

The elastic beam or plate theory (Obert, Duvall, and Merill, 1960; and Wright, 1973), nonelastic analysis (Haycocks, Townsend, and Lucas, 1974; and Haycocks, 1976), and rock mass classifications have been used in designing roof spans in mines and tunnels for some years (Haycocks, 1979; and Bieniawski,1984).

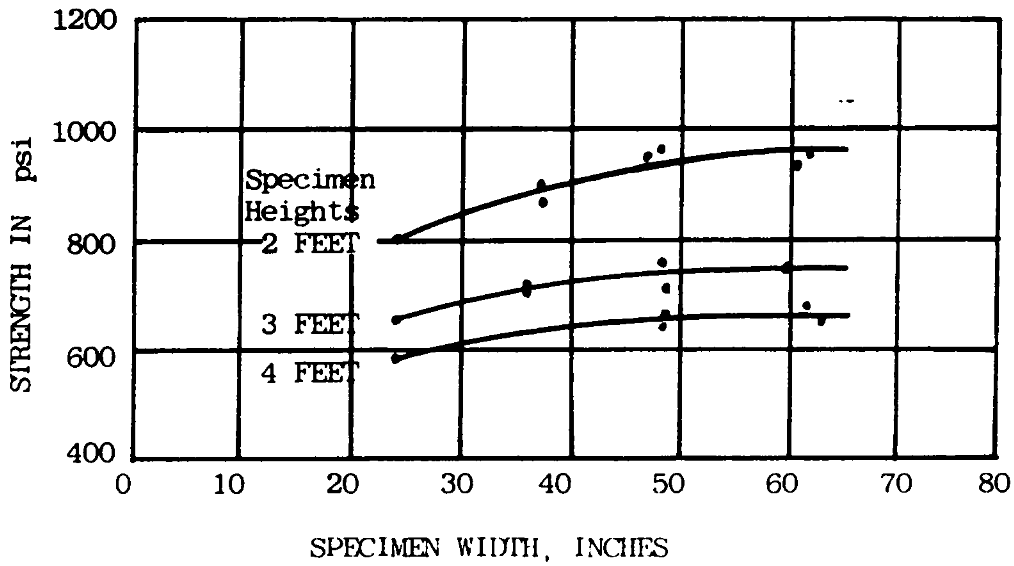
Rock mass classifications are the empirical design approaches. There are many different rock classification systems in existence today and the more common ones are listed as follows (Bieniawski, 1984):

Name of classification	Originator and date	Applications
Rock loads	- Terzaghi, 1946	- for tunnels
Stand-up time	- Lauffer, 1958	- for tunnels
RQD	- Deere, 1964	- Core logging
Intact rock	- Deere & Miller	- Communication



Relationship between strength and cube size of large and medium coal specimens tested underground, (After Bieniawski)

Figure 2.5



Relationship between strength and width of square coal specimens tested in-situ, (After Bieniawski)

Figure 2.6

strength	1966	
RSR concept	- Wickham, et al. 1972	- for tunnels
Geomechanics Classification (RMR system)	- Bieniawski, 1973	- for tunnels, mines, & foundations
Q-system	- Barton, et al. 1974	- for tunnels,& large chambers
Strength/block size	- Franklin, 1975	- for tunnels
Basic geotechnical classification	- ISRM, 1981	- for general purpose.

The Geomechanics Classification proposed by Bieniawski (1980, 1981, 1982) has been used in classifying roof conditions and determining unsupported roof spans and stand-up time for a roof condition. The classification is based on the following parameters:

-- uniaxial compressive strength of rock material

-- rock quality designation RQD

-- spacing of discontinuities

-- condition of discontinuities

-- orientation of discontinuities

-- groundwater conditions

All these parameters are determined and importance ratings are assigned to each geomechanic parameter (Table 2.3). Five parameters are grouped into five ranges of values. Then the ratings for the five parameters are summed up to obtain the basic rock mass rating (RMR). In the next step, the effect of the 'strike and dip of discontinuities' is included by adjusting the basic rock mass rating (Table 2.3, Section B). The value of "discontinuity orientation" is given by qualitative descriptions such as "favorable" or "fair." References are made to the bottom section of Table 2.3 to determine whether strike and dip orientations are favorable. After the adjustment for discontinuity orientations has been made, the rock mass is classified according to section C of Table 2.3 which groups the final (adjusted) rock mass ratings into five rock mass classes. Each rock mass class is in a group of 20 ratings and gives a value of roof span and corresponding average stand-up time (Section D, Table 2.3). Figure 2.7 depicts the relationship between the stand-up time and the roof span for various rock mass ratings (Bieniawski, 1981).

Geomechanics Classification facilitates the estimation of a safe roof span, and when a roof span is determined using other considerations, it enables the estimation of stand-up time.

Smith (1984) investigated the mine roof fall characteristics of 250 falls in 5 different room-and-pillar

TABLE 2.3

Geomechanics Classification of Bieniawski (1984)

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS

	PARAMETER		RANGES OF VALUES						
1	Strength of intact rock material	Point-load strength index	> 10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range uniaxial compressive test is preferred		
		Uniaxial compressive strength	>250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5-25 MPa	1-5 MPa	< 1 MPa
	Rating		15	12	7	4	2	1	0
2	Drill core quality RQD		90% - 100%	75% - 90%	50% - 75%	25% - 50%	< 25%		
	Rating		20	17	13	8	3		
3	Spacing of discontinuities		>2 m	0.6 - 2 m	200 - 600 mm	60 - 200 mm	< 60 mm		
	Rating		20	15	10	8	5		
4	Condition of discontinuities		Very rough surfaces Not continuous No seperation Unweathered wall rock	Slightly rough surfaces Separation < 1 mm Slightly weathered walls	Slightly rough surfaces Seperation < 1 mm Highly weathered walls	Slickensided surfaces OR Gouge < 5 mm thick OR Separation 1-5 mm Continuous	Soft gouge > 5 mm thick OR Separation > 5 mm Continous		
	Rating		30	25	20	10	0		
5	Ground water	Inflow per 10 m tunnel length	None	< 10 litres/min	10-25 litres/min	25 - 125 litres/min	> 125		
		OR Ratio $\frac{\text{joint water pressure}}{\text{major principal stress}}$	0	0.0-0.1	0.1-0.2	0.2-0.5	> 0.5		
		OR General conditions	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		15	10	7	4	0		

TABLE 2.3 Continued

B. RATING ADJUSTMENT FOR JOINT ORIENTATIONS

Strike and dip orientations of joints		Very favourable	Favourable	Fair	Unfavourable	Very unfavourable
Ratings	Tunnels	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	-5	-25	-50	-60

C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS

Rating	100 ← 81	80 ← 61	60 ← 41	40 ← 21	< 20
Class No	I	II	III	IV	V
Description	Very good rock	Good rock	Fair rock	Poor rock	Very poor rock

D. MEANING OF ROCK MASS CLASSES

Class No	I	II	III	IV	V
Average stand-up time	10 years for 15 m span	6 months for 8 m span	1 week for 5 m span	10 hours for 2.5 m span	30 minutes for 1 m span
Cohesion of the rock mass	> 400 kPa	300 - 400 kPa	200 - 300 kPa	100 - 200 kPa	< 100 kPa
Friction angle of the rock mass	< 45°	35° - 45°	25° - 35°	15° - 25°	< 15°

TABLE 2.3 Continued

Strike perpendicular to tunnel axis			
Drive with dip		Drive against dip	
Dip 45°–90°	Dip 20°–45°	Dip 45°–90°	Dip 20°–45°
Very favorable	Favorable	Fair	Unfavorable

Strike parallel to tunnel axis	
Dip 20°–45°	Dip 45°–90°
Fair	Very unfavorable

Irrespective of strike
Dip 0°–20°
Fair

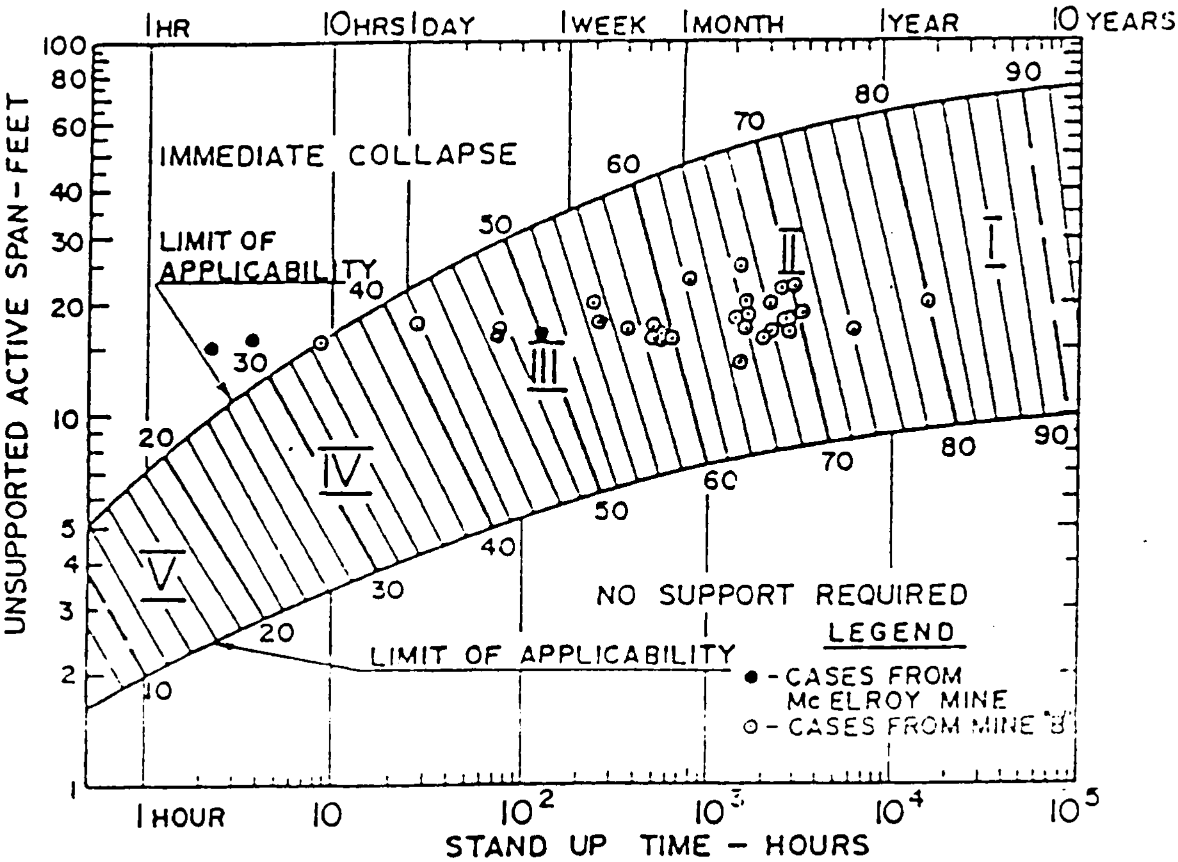
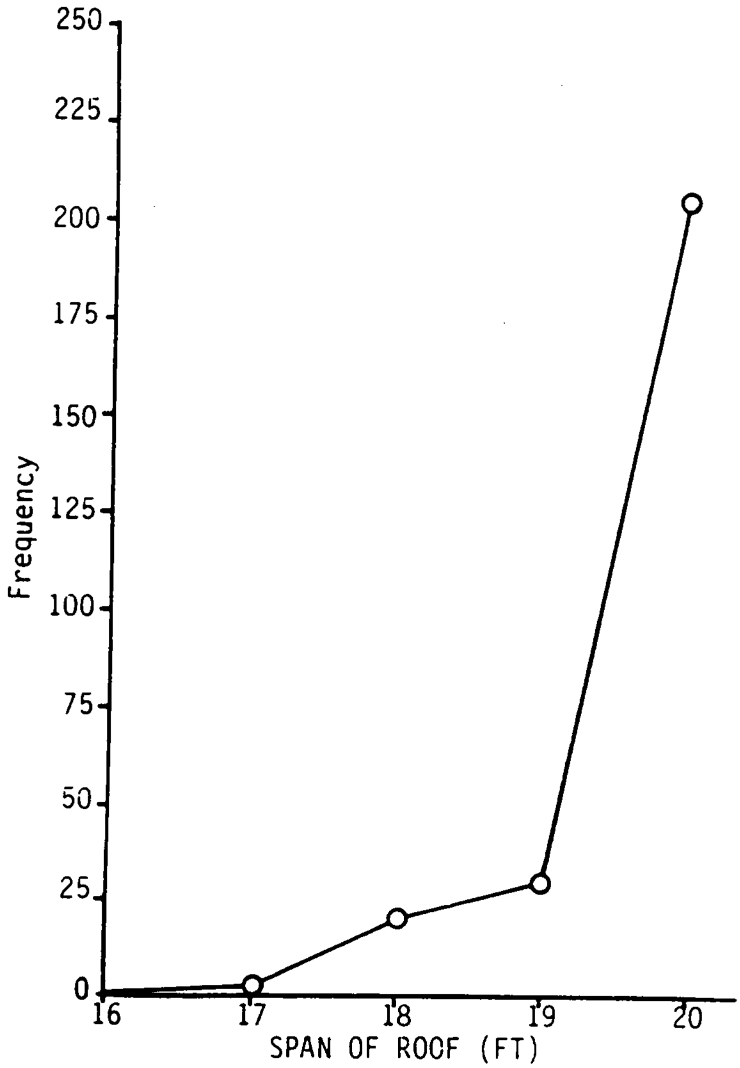


Fig.2.7 Stand-up time versus roof span for coal mine roofs.
(After Bieniawski, 1982)

coal mines, located in Pike, Martin, and Floyd counties of Eastern Kentucky, to determine the relationship of selected parameters associated with roof failure and the assumed condition of the mine roof before failure. The relationship was tested, utilizing multiple linear regression techniques, on the selected parameters and mine roof conditions. The results indicated that roofs supported by posts or cribs very rarely failed, whereas most of the roof falls were in the area supported by mechanical-anchor bolts and at the intersections of mine entries. The span of entries associated with mine roof falls was an interesting parameter in that study. There were no roof falls in the entries which had a roof span of less than or equal to 16 feet. As the roof span of entries increased, the more number of roof falls in those entries increased. The rate of increase of the number of mine roof falls was maximum when the roof span increased beyond 20 feet, and it was lowest in the 16 to 17 feet range. The results can be used to determine the optimum roof span of entries (Figure 2.8).

2.3.2 STRENGTH OF COAL PILLARS

The strength of coal pillars in a mine is determined by many factors, most of which are beyond the control of the mining engineer. Early approaches at pillar strength determination are significantly different from more modern



Distribution of span of entries associated with mine roof falls. (After Smith, 1984)

Figure 2.8

equations.

Earlier Approaches

Three pillar design formulae considered to be capable of covering general applications have been selected for discussion (Holland, 1973).

(i) Holland-Gaddy formula (1956, 1962, 1964): This formula is most frequently used for coal pillar design.

$$S = \frac{K\sqrt{L}}{T}$$

and, $K = S_p\sqrt{D}$

where, S is strength of pillars in psi,

L is lateral dimension of pillar in inches,

T is thickness in inches,

S_p is strength of a cubical specimen of coal of dimension D,

K is the coefficient depending upon the coal quality.

(ii) Morrison-Corlett-Rice formula:

$S = K\sqrt{W/T}$.

where, S = strength of pillar in psi,

W = least width

T = thickness

K = crushing strength of a 1 inch cube specimen.

(iii) Obert's formula: This is limited to the design of rib pillars and hard rock pillars.

$$S_p = C_1(0.778 + 0.222\ W_p/H_p)$$

where, S_p = strength of pillar in psi,

C_1 = average strength in psi of cubical coal specimens,

W_p = least width of pillar,

H_p = height of pillar.

In using these formulae, all should be modified by a factor of safety. The safety factor runs from 1.7 to about 2 in the Holland Gaddy formula, 4 or 5 in the Morrison-Corlett-Rice formula and 4.5 in Obert's formula.

Limitations of these Formulae: If the height of the coal pillar is more than the pillar's least dimension, or the L/T ratio is more than 12, the Holland-Gaddy cannot be used. In situations where the L/T ratio is less than 1 or the width-over-height ratio is more than 8, the Morrison-Corlett-Rice formula should not be used.

Recent Approaches

The most recent work on pillar design has been done by Bieniawski (1968), Salamon (1967) and Protodyakonov (1964).

(i) Bieniawski tested different sizes of coal samples in-situ. His in-situ experiments were confined to Witbank coal only. He observed that the strength did not vary for sizes below about 2.5 inches, which was the least distance between discontinuities for Witbank coal. There was variation in strength between 2.5 inches and 5 feet cube. When the size is increased beyond 5 feet the strength did not change. These results are shown in Figures 2.5 and 2.6. On the basis of these results, Bieniawski (1968) took the 5 feet cube as the basis of his calculations and arrived at the following formula:

$$\sigma = \sigma_c (0.645 + 0.355\ w/h)$$

where, σ_c is the compressive strength in psi for 5 feet cube.

w is the width of the pillar in feet.

h is the working height in feet.

For Witbank coal, the equation can be written as follows:

$$\sigma = 400 + 220\ w/h.$$

For pillar design Bieniawski's approach must be treated with caution since his studies were confined to only one mine (Ali, Raju, and Singh, 1971). Skelly (1977) conducted an underground pillar strength test in southern West Virginia and found that constants in the Bieniawski's equation change to:

$$\sigma = \sigma_c (.78 + 0.22\ w/h).$$

(ii) Salamon (1967) studied 98 cases of stable and 27 cases of collapsed pillars of different South African Coal seams and deduced a statistical approach to designing mine pillars for new mines and checking the stability of old workings. He derived this approach on a purely statistical basis taking the difference in coal strength for different coalfields as a random variation. This is reasonable for South African coalfields because the strength variation for different coals does not exceed 12.5 percent (Sheorey, 1970). Salamon arrived at the following formula assuming that the cause of failure was super incumbent load,

$$\sigma = 1320 \frac{w^{0.46}}{h^{0.66}}$$

This formula can be applied to coal pillars in American mines willing to accept approximate results because the compressive strength for American coals varies from 1150 psi to about 7000 psi. It is possible to develop a similar strength formula for American coals and apply it to all the coalfields of the United States. Different strength groups have also been picked up during tests for all U.S. coals (Bieniawski, 1982) and a formula for each of the groups can be derived by Salamon's approach.

Salamon's studies on South African coal were carried out on actual pillars greater than 5 feet. His formula for pillar design holds for pillars greater than 5 feet. Thus Bieniawski's conclusion appears doubtful since Salamon's formula indicates a strength variation after 5 feet in size.

(iii) Protodyakonov (1964) deduced the following expression for pillar size effect:

$$\sigma_d = \frac{d + mb}{d + b} \cdot \sigma_m$$

and, $$m = \frac{\sigma_o}{\sigma_m}$$

where, σ_d is the strength of a cubic rock specimen with side d.

σ_o is the strength of a cubic rock specimen with $d = 0$.

σ_m is the strength of a cubic rock specimen with $d = \infty$

b is the distance between discontinuities.

If the effect of height of working and width of coal pillars is taken in account, then the modified equation is,

$$\sigma_d = \frac{w + mb}{w + b} \sigma_m (a + c.\frac{h}{w})$$

in which a and c are empirical constants and can be determined in the laboratory for coal under investigation.

PILLAR LOAD

The pillar load is measured as the average pressure acting on the pillar. Since the weight of the overburden increases by approximately 1.1 lbs/square inch for every foot below the surface, the average pillar load is considered as:

$$p = \frac{1.1\,H}{1 - e}$$

and,

$$e = 1 - \left(\frac{w}{w+B}\right)^2$$

where, e - extraction ratio

H - depth of coal seam

B - width of room or entry

w = width of square pillars

Duvall and Denkhaus (1962) have found the influence of percentage extraction on pillar loading which is demonstrated in Figure 2.9.

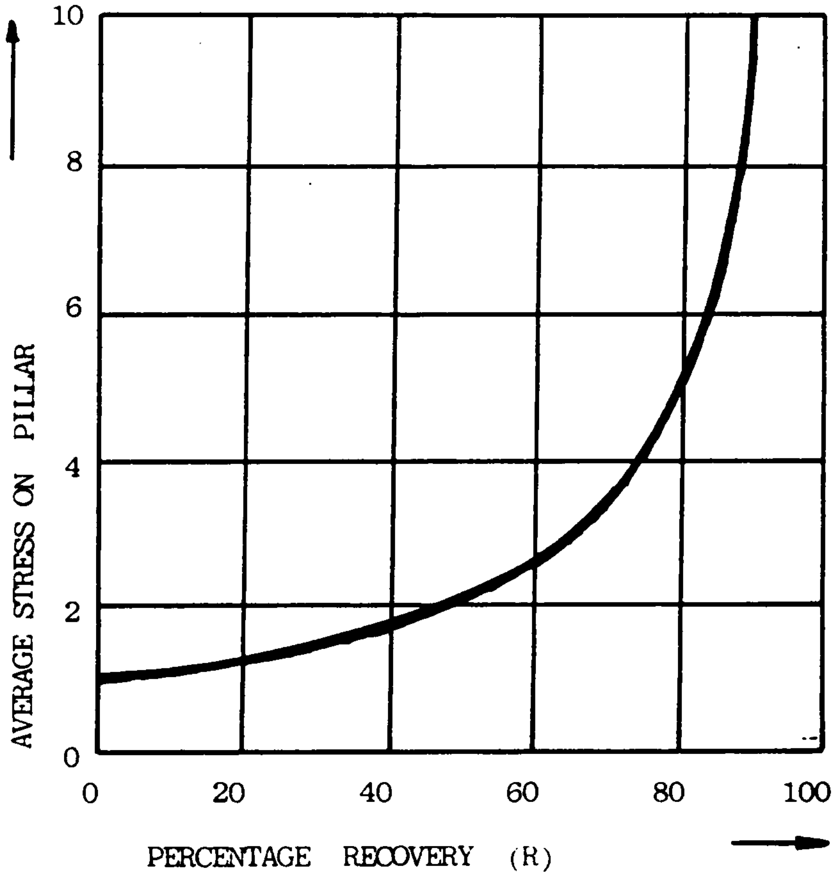
SAFETY FACTOR

The safety factor is defined as the ratio of strength to load.

$$S = \frac{\text{Strength}}{\text{Load}}$$

There is a 50% probability of failure when safety factor = 1. However, if the safety factor is greater than one the structure is defined as stable and there is less than 50% probability of failure. Higher safety factors result in larger pillar sizes, consequently increasing haulage or production costs (Haycocks, et al. 1984).

Salamon and Munro (1967) calculated safety factors on the basis of the data obtained in the course of the survey and found that collapsed pillar structures were clustered around safety factor = 1, where the chances of failure and stability are equal. The stable pillar structures were



Influence of percentage extraction on pillar loading (After Denkhaus)

Figure 2.9

distributed over a wide range, usually larger than unity and concentrated between safety factors 1.3 and 1.9. The optimal value of the safety factor lay in the range where 50 percent of the stable cases were most densely concentrated. The range and its mean could be used as a guide when new workings are designed or old ones are re-examined. However, the final decision, with regard to the selection of safety factors, must take into consideration local mining experience (Salamon, 1967).

The survey conducted by Bieniawski (1982) for deciding on safety factors for U.S. room-and-pillar coal mining provides information for different pillar design formulae. The safety factors ranging from 1.5 to 2.0 are appropriate for U.S. coal mining conditions using Bieniawski's Formula. For the Holland Formula it has been recommended to use a safety factor of 2.0 as an average, 1.8 as a minimum, and 2.2 as the maximum. The Obert-Duvall formula had a factor of up to 4 suggested if no size effect is included.

PROBABILISTIC APPROACH OF DESIGNING COAL PILLARS

Computation of underground coal pillar strength requires the estimation of compressive or crushing strength of cubical coal specimens. The strength of a cubical coal specimen can not be predicted with certainty due to variations in coal structure and composition. An alternative to using the average or a single conservative value for strength is to recognize, measure and include variability in the determination of coal pillar strength. Variation in coal strength for coal seams has been reported by many investigators.

Davies (1977) collected standard compressive strength test data from all parts of the British coalfield and found the mean value and standard deviation to be 30.4 MN/metre^2 and 12.4 MN/metre^2, respectively. The minimum value was 11.9 MN/metre^2 and the maximum was 55.0 MN/metre^2. The compressive strengths for identical coal samples and testing conditions varied from one sample to another.

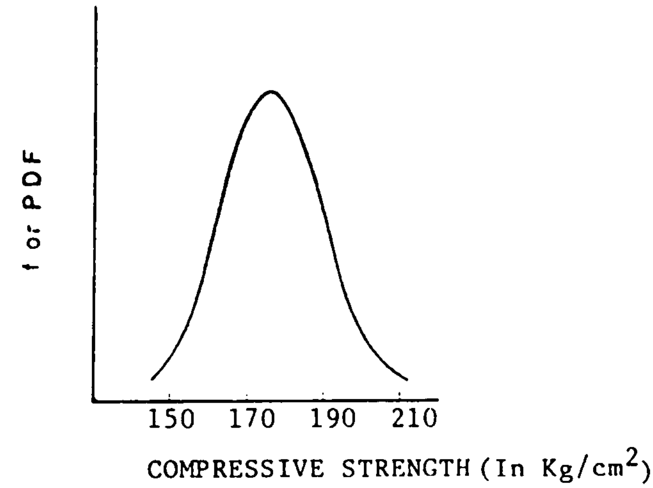
Illinois coal (Herrin # 6) was tested and the compressive strength varied from 102 Kg/cm^2 to 236 Kg/cm^2. The average value and standard deviation were 157 and 39 Kg/cm^2, respectively (Chugh, et al. 1981).

As indicated in the examples above, the compressive strength of cubical coal specimen is a variable. To be realistic, probability should be incorporated into the strength determination or coal pillar design procedure to describe the inherent variability associated with the compressive strength value of coal. The variation in coal specimen strength is random variation (Canmet, 1977).

If the probability distribution of a random variable is identified, it is possible to compute the probability of occurrence of an event. Figure 2.10 is a hypothetical probability density function plot of compressive strengths of identical cubical coal specimens. The distribution can be either symmetric or asymmetric. Most of the compressive strengths values are distributed around the median as depicted in the example case in Figure 2.11, which is an asymmetric distribution. The probability of occurrence of compressive strengths equal to or less than S_1 will be the summation of the area under the probability density curve up to S_1 value.

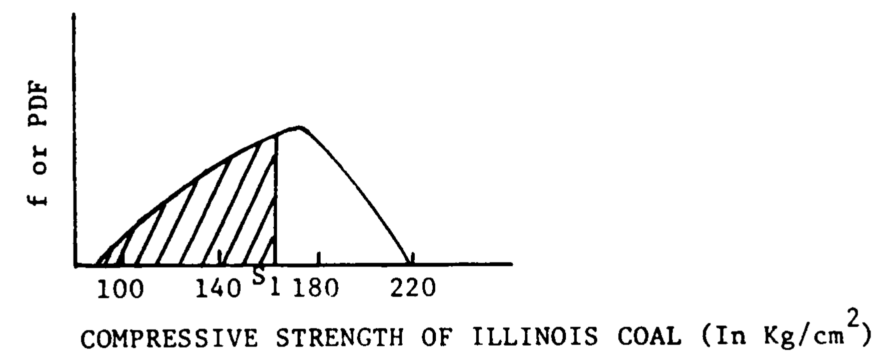
The factor of safety,FS, is related to strength and for a given roof load the probability of occurrence of safety factors equal to or less than FS_1 will be same as the probability of occurrence of compressive strengths equal to or less than S_1. The probabilty of ocurrence of safety factors greater than FS_1 will be the summation of the area under the probability curve on the right side of FS_1.

If the lowest value of compressive strength of cubical coal specimens is selected from the probability density curve for the determination of safety factor greater than unity, the coal pillars will be stable with probability approaching 100%. If the largest value of compressive strength is used to compute safety factor greater than unity by a small increment, then failure probably will occur or pillars will collapse with probability approaching 100%.



Probability Density Function versus
Compressive Strength of Coal.

Figure 2.10



Probability Density Function versus
Compressive Strength of Illinois Coal.

Figure 2.11

It is possible to compute the probability of coal pillar's stability condition if the probability distribution of compressive strength, the roof load and the desired safety factor are known.

If sufficient coal strength test data are available, the concepts of probability and Monte Carlo simulation are used to generate coal strength value. The probability distributions useful in generating coal strength, shape characteristics of probability distribution, identification of the distribution of a random variable, estimation of the parameters, testing the probability distribution, and generation of random variables are discussed in Appendix F.

The cumulative distribution function (c.d.f.) is derived after first identifying the probability density function (p.d.f.) of the coal strength. While the coal strength test data collected would include only finite values for strength, the probability density function (p.d.f.) fit to the data would place no upper or lower limit on the value of this random variable although the probability of occurence of extremely large or small values of the coal strength is quite small.

2.3.3 IMPROVED METHOD OF LAYOUT DESIGN

The pillar design procedure discussed previously considers only the pillar stability. In addition to this factor, layouts should be designed to achieve the highest possible productivity and percentage coal extraction. The pillar design formulae imply that stability considerations keep the working height to a minimum and the pillar width to a maximum. But high productivity and coal extraction is achieved when working height is maximum and the pillar width minimum (mathematical model and computer simulation results; Haycocks, et al. 1984). The productivity of a room-and-pillar mining section is measured by the amount of coal produced during a given time period. The friendly room-and-pillar mining simulator (FRAPS), which is a stochastic face mining simulator with probabilistic options, is used to predict the productivity (Haycocks, et al. 1984).

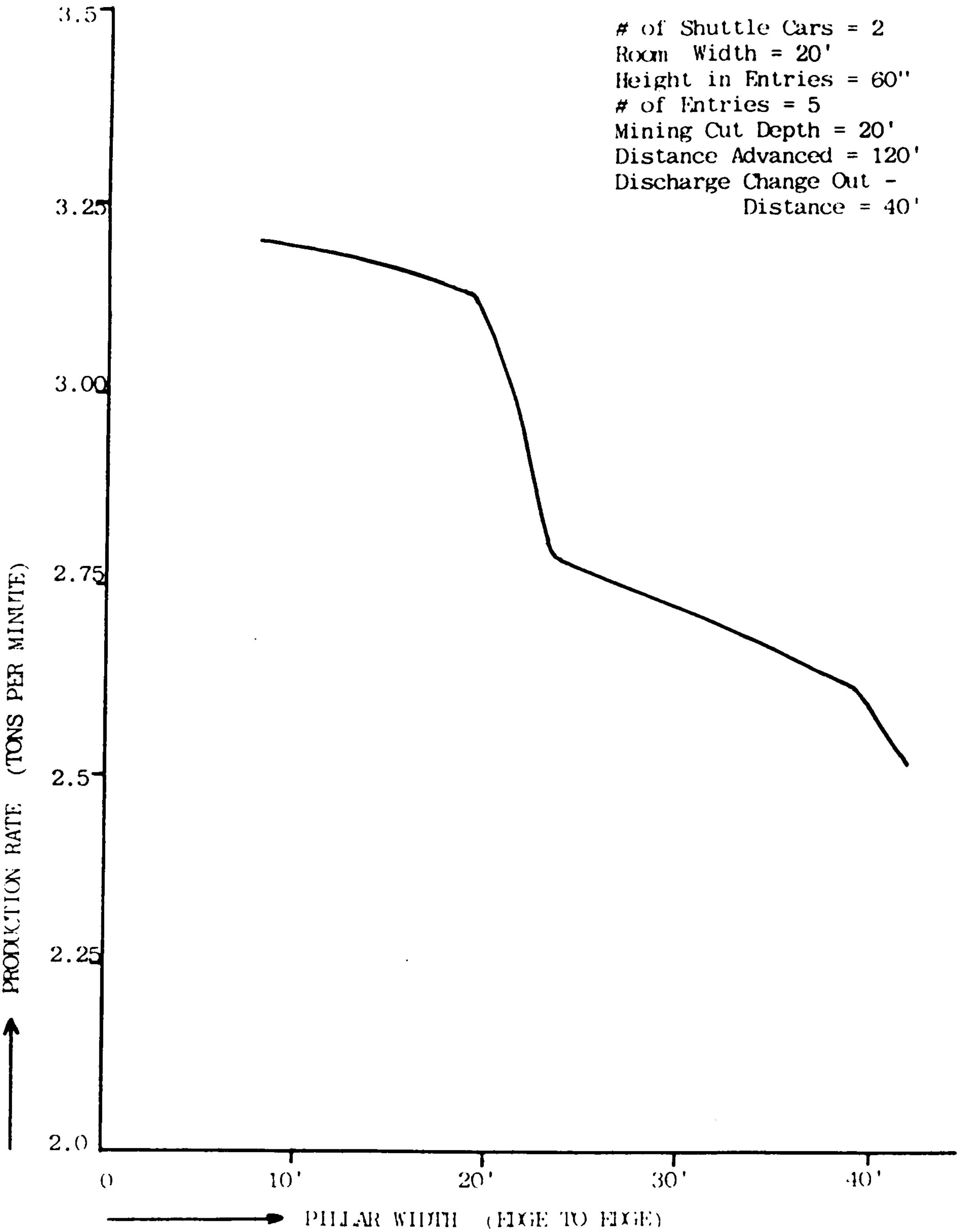
All operations involved in coal production, such as cutting, drilling, blasting, loading, and roof supporting, and various interruptions or delays due to ground control problems, equipment repairs, and methane dilution, are simulated with their inherent variability, and the time required to carry out the operations, together with the amount of coal produced, is recorded. All operations are subdivided into smallest elements and then they are simulated. The time required to perform each of these elemental tasks is studied. FRAPS also offers a means of determining how the number of entries, entry and breakthrough center distances, etc., contribute to the overall efficiency of the production unit in question. Alternate systems can be simulated and compared to determine the mining plan with optimum production potential. These mining dimensions can be evaluated in conjunction with the rock mechanics work discussed earlier to insure that the pillars of the proposed mining layout will be stable and safe (Figure 2.12).

2.3.4 DESIGN OF TOTAL EXTRACTION LAYOUTS

This section evaluates procedures for design of panel dimensions, interpanel pillars, roof control in total extraction panels, and roof control in pillar extraction systems.

Design of Panel Dimensions

The layout of total extraction panels is designed to ensure safe and efficient mining conditions in all structural regions of the mine. Safe and efficient mining conditions require avoiding high stress concentrations at the abutments of total extraction panels, which can be achieved by designing panel spans in such a way that the upper roof strata above the extracted panel caves to surface (Fauconnier, 1982). In cases in which this is neither possible nor practicable, the panel spans have to be limited and total extraction panels will have to be separated by substantial barrier pillars to minimize the



Relationship between Pillar Width and Production Rate (Tons/Minute)

Figure 2.12

effects of interaction between panels.

The composition of the upper roof strata is a most important factor affecting the design of a total extraction panel. Two different types of roof strata conditions in the coalfields are as follows (Kresten, 1982):

1. a generally weak and incompetent upper roof comprising shale and weak sandstone layers, and
2. a roof strata which contains massive beds of competent sandstone.

<u>Incompetent Roof Strata</u>: - If the roof strata is weak and incompetent, then high stresses occurring in the vicinity of abutments frequently are enough to cause failure of the roof strata. The stress-induced fractures tend to develop in the direction of the near vertical maximum compressive stress. Because the lower roof strata in the mined out area are in tension, gravitational collapse of the intensely fractured roof strata can take place readily and caving of the roof is continuous and extends to the surface. The maximum compressive stresses at the abutments are reached when the mining span is large enough for caving of the roof strata to have extended to the surface. The critical mining span, L, at which the maximum compressive stresses at the abutments are reached, is given by (Fauconnier & Kersten, 1982):

$$L = 2HCotB$$

where, H is depth of mining below surface

B is average of dip angle of face induced fractures

Competent Roof Strata: - Salamon (1972) developed a model to calculate the maximum span that can be supported by a massive bed of competent sandstone. According to Salamon, the maximum stress in a uniformly loaded, infinitely long rectangular plate is proportional to the intensity of the load, the square of the span, S, and the inverse of the square of thickness of plate, t.

$$\sigma_{max} = k\ D\ \left(\frac{S}{t}\right)^2$$

where, D is depth below surface of the sandstone bed,

k is coefficient.

Design of Inter-Panel Pillars

The inter-panel pillars are also called barrier pillars and, in the case of total extraction systems, serve three functions:

1. to protect the mine against flooding from large bodies of water and act as ventilation barrier
2. to isolate individual extraction panels
3. to protect entry pillars from being subjected to the high stress area that usually surrounds mined out areas.

Two opposing criteria are considered in the design of inter-panel pillars:

1. the inter-panel pillar should be as narrow as possible to minimize loss of coal.
2. the inter-panels pillar should be as wide necessary to protect the development entries and panel operations from the effects of high abutment stresses.

Strength of Inter-Panel or Barrier Pillars

The design of inter-panel pillars has been done in the past on the basis of judgement and experience of the mining engineers concerned or by one of the three formulae discussed below.

(i) Ashley or Mine Inspector's formula:

$$W = 20 + 4T + 0.1D$$

where, W = width of pillar in feet

T = thickness of pillar in feet

D = thickness of the overburden in feet

(ii) Holland's formula:

$$D = 15T, \qquad \text{or,} \quad D = 5\left(\frac{\log 2\ W_2}{0.09\ \log e}\right)$$

where, T = thickness of pillar in feet,

D = width of barrier pillars in feet,

W_2 = the estimated convergence on the high-stress side of the pillar in millimeters

(iii) The third formula is based on the ratio of pillar area to pillar circumference.

$$W_{eff} = \frac{4A}{C}$$

where, A = pillar area

C = pillar circumference

W_{eff} = pillar width

The average pillar stress, p_m, can be estimated using the classical tributary area theory

$$p_m = \frac{\gamma H}{1-e}$$

where, γ is the weight of rock per unit volume,

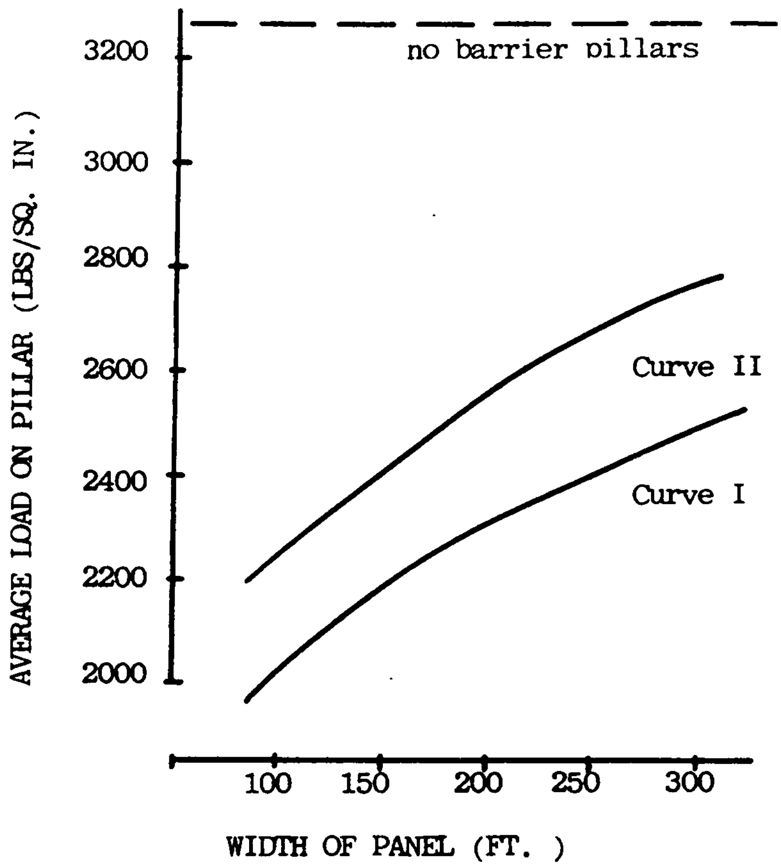
H is the depth of seam below surface,

e is the extraction ratio.

Starfield (1968) studied the pillar loads variation characteristics in room-and-pillar mining development panels. The pillars which are closest to the barriers carry the least load while the pillars near the center of the panel carry the greatest load. The average pressure on a pillar near the center of the panel versus the panel widths relationship is depicted in Figure 2.13. Curve I shows the pressures when barrier pillars are effectively of infinite width while curve II shows the pressure reached when a large number of panels have been mined. In both cases, pillar load reduces in the proximity of barriers.

2.3.5 ROOF CONTROL IN PILLAR EXTRACTION SYSTEMS

The roof support system used in pillar extraction



The average load on the pillars near the centre of a panel.
Curve I: One panel only.
Curve II:A large no. of panels separated by 100ft. barriers
(After Starfield)

Figure 2.13

methods must be designed to protect men and equipment against the effects of local rooffalls as well as periodic roof strata failures. Control of the immediate roof in total extraction panels is far more complex than room-and-pillar partial extraction mining and depends on the following factors (Fauconnier, 1982):

- strength properties of the immediate and upper roof,
- strength properties of the coal seam, and working height,
- depth of mining below surface,
- panel geometry,
- rate at which the working face is advanced,
- sequence of pillar extraction,
- sequence of cuts in a pillar.

The strength properties and degree of lamination of the immediate roof determine the minimum span that can be left unsupported. A weak and highly laminated roof tends to cave at the working face. Strong massive sandstone beds in the immediate roof can make it impossible to employ pillar extraction methods because it does not cave regularly.

The sequence of pillar extraction is also important. The diagonal extraction line produces steeply dipping fractures which tend to intersect the rooms or entries parallel to the overall pillar extraction line. According to Fauconnier and Kresten, the orientation of the stress-induced fractures is somewhat more complex close to the barrier pillars:

- the strike direction of the fractures that, in the center of the panel, is parallel to the pillar extraction line, rotates towards the direction of the barrier pillars,
- the angle of dip of these fractures tends to flatten in the vicinity of the barrier pillars.

The sequence of cuts during the extraction of a pillar is important for controlling the local roof strata. As it requires a knowledge of the local stiffness of the mining layout and post failure behavior of the pillar, it is most difficult to assess.

A roof support with high initial stiffness can provide safe working conditions in the vicinity of the pillar edge. Timber supports are best for this purpose. The use of wedges with timber supports increases the total yield range of timber even further.

It is simple to install timber supports and they offer relatively high resistance with small compression. Timber posts and cribs are easily available in every coalfield and their performance, suitability and costs are familiar to most people in mining industry. Most of the timber posts are a section of the trunk of a tree with diameters from 4 inches upward (a six inch post is most common). The length-to-diameter ratio should be less than or equal to 15 for wood posts (Faulkner and Yu, 1985). The strength of timber post for a certain species of wood is a function of several variables such that (Suddarth and Woeste, 1977):

$$p = f(m, i, g, r)$$

where:

p = strength of post,

m = moisture content in the post,

i = imperfections such as knots,

g = specific gravity,

r = length to diameter ratio.

Faulkner (1985) performed several tests and tried to find this relation through regression. According to Faulkner and Yu (1985), the species of wood has a great effect on the ultimate strength of a post. The hardwoods such as oak, poplar, hickory, maple and locust exhibit much higher compressive strengths than softwoods such as pine and spruce. Dry wood of low moisture content is much stronger than "green" wood, which has high moisture content.

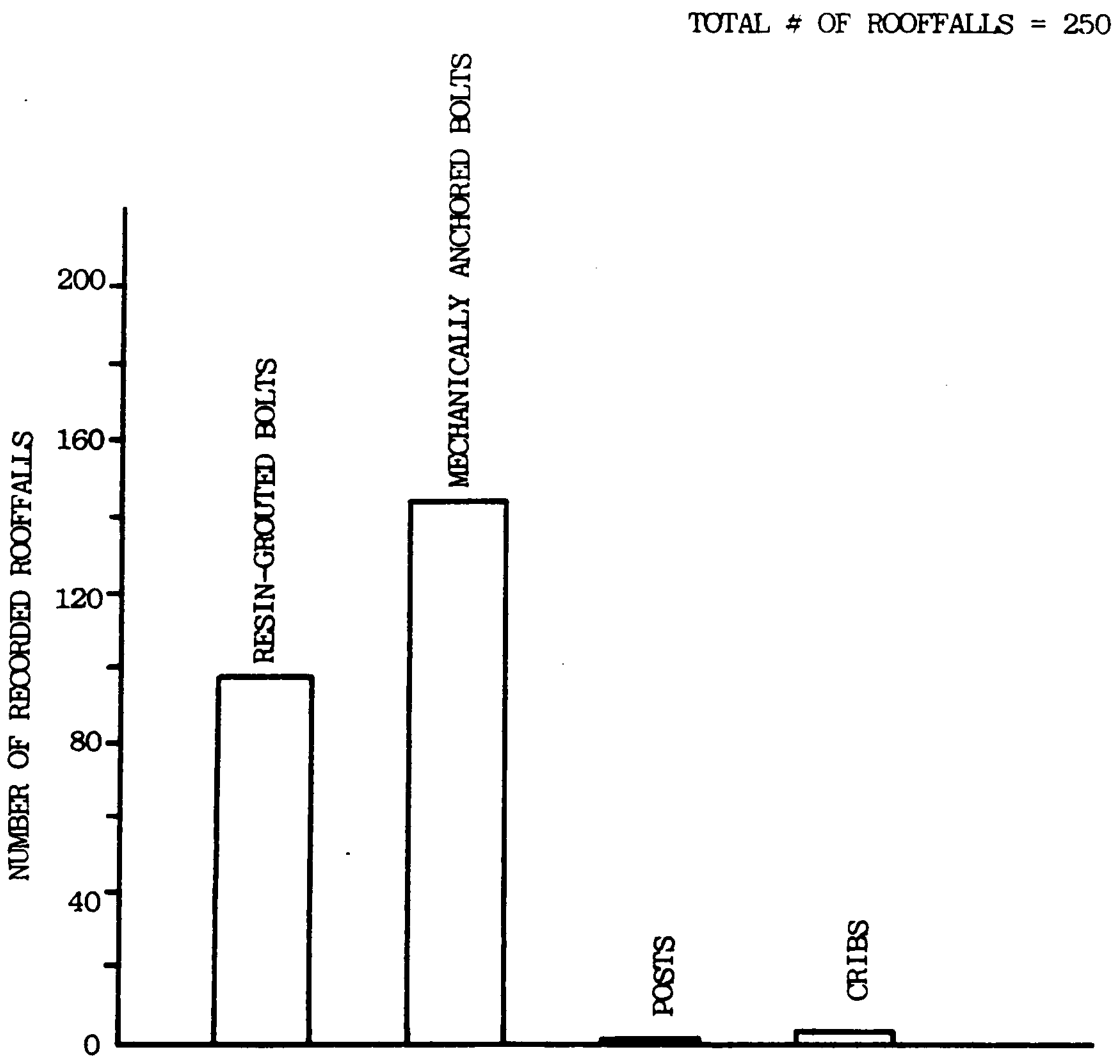
The load-bearing and yield characterstics of posts can be improved by using wedges between the post and the mine roof which not only tighten post installation but allow additional yield as they are compressed perpendicular to the grain of the wood. Caps and headers also improve the yieldability of a post in a similar way.

There are several kinds of timber props in pillar extraction panels. Hepplewhite, Kirkby, and tri-set props are most common (Faulkner and Yu, 1985). The Hepplewhite prop is produced by tapering the lower end of a timber post. This prop exhibits improved compression and yield characteristics in conjunction with a headboard. The Kirkby prop consists of a light steel tube with reinforcing bands on the ends. Various combinations of wooden plugs can be inserted into the tube to get the desired yield performance. The "tri-set" consists of three timber posts working together. It is slightly less strong than three times the strength of a single post, deforms about the same amount before failure, and is more stable as deformation continues.

Cribbing is the use of flat timbers laid at right angles on top of each other to support heavy roof loads in pillar extraction panels. Cribs exhibit good load - deformation characterstics. Wood cribs handle roof support demands that cannot be met by wooden props. Cribs can be square, rectangular or triangular in shape. They can be open or solid depending on the way the timbers are arranged and cut. Hollow centers of cribs may be filled with waste rock to further strengthen them. The yieldability of the wood crib is advantageous since the load on a crib is perpendicular to the grain of the wood. A crib yields in a stable manner as the load increases.

Roof bolts are also used in the splits to support roof. They can be installed rapidly and do not interfere with the movement of machinery. The best designed support systems for a total extraction panel are those that employ bolts or timbers installed at specific intervals with an increased density at roadway intersections. A roof fall survey by Smith (1984) indicates that areas supported by posts and cribs have the least number of rooffalls and the result is depicted in Figure 2.14.

The Bureau of Mines has developed a mobile roof support system for retreat mining panels which would be placed and retrieved remotely without danger to the operator (Thompson, 1984). It was designed, built, and field tested in a Utah coal mine (Figure 2.15). The mobile roof support was remotely operated, battery powered, and rubber-tired. The machine carries four jacks, two on the body of the machine, and two at the end of hinged arms. The jacks extend to form columns between the floor and roof, each with 30 tons of potential support. The jacks are hydraulically locked, and the load is distributed to three points on each jack, without loading the machine chasis



DISTRIBUTION OF TYPE OF ROOF SUPPORT ASSOCIATED WITH FALLS
(After Smith, 1984)
Figure 2.14

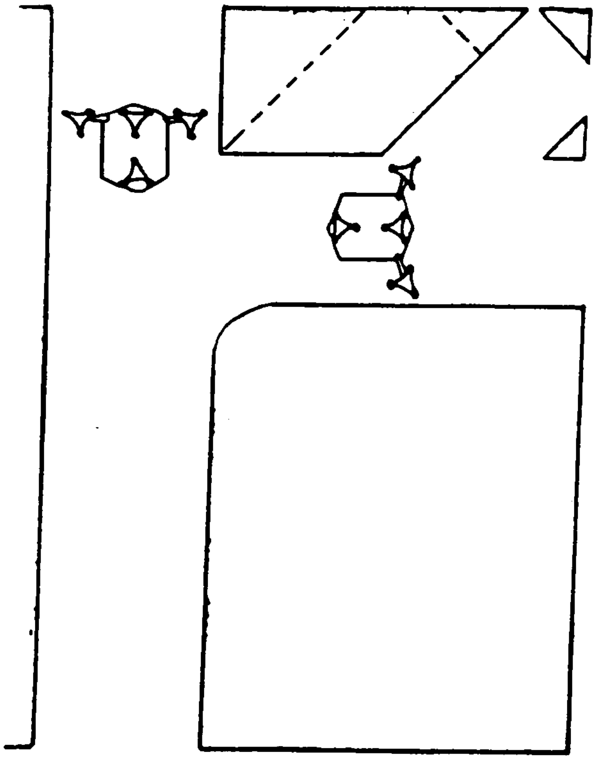


FIGURE 2.15: Ground control plan with mobile remote roof support (remotely operated).
(After Thompson, 1984)

(Thompson, 1984). This system provides added safety for coal miners by eliminating the need to work in a hazardous area setting timber posts and cribs, and increases productivity because of fewer manual support setting operations.

Results of the field test were encouraging and the concept was accepted by mine management, the miners and several industry personnel who witnessed the field trials.

2.3.6 EFFECTS OF RETREAT MINING ON GROUND SURFACE

Retreat mining or total extraction disturbs the natural equilibrium of rock mass in the vicinity of the extractions, changing the distribution of loads in the medium and causing horizontal and vertical displacements. These displacements propagate from the mine opening, through the overlying strata to the surface, and result in surface subsidence (Karmis, Haycocks and Triplett, 1984). Depending on the extent of the workings, their depth below surface, and the physical properties of the rock mass, displacements on the ground surface can be manifested in two principal modes of ground settlement, sinkhole, and trough subsidence. Sinkholes are characterized by a sudden and sometimes violent collapse of the surface and usually occur above shallow mines with incompetent overburden. The surface expression of trough subsidence is the formation of a three-dimensional subsidence basin (Karmis, Haycocks, Webb, and Triplett, 1981).

Surface subsidence is an important environmental problem of retreat mining. Damages due to subsidence range from land settlement to severe structural damage in rural as well as urban area (Karmis, Goodman, and Hasenfus, 1984). The federal government estimates that each year underground coal mining causes $25 million to $35 million in subsidence damage to residences, schools and commercial/industrial buildings in the United States, and $3 million to $4 million in damage to roads, utilities and services (Bruhn, McCann, Speck, and Gray, 1982). Much of the damage occurs in Pennsylvania, where the concentration of houses and other structures above active mine areas is greater than anywhere else in the United States. With similar damage occurring above operating mines in West Virginia, Ohio, Illinois, Virginia and other coal producing states, subsidence effects on structures is a topic worthy of attention (Bruhn, et al. 1982).

The serious problems resulting from surface subsidence are due to differential movements at various points of a structure. The tilt, the curvature, the horizontal strain, and horizontal distortion are the important components of differential movement.

The tilt at a point is the slope of the subsidence curve. Tilt of the surface may make the tall buildings unstable. Curvature is the difference in tilt between two neighboring points. This causes the formation of a concave or convex area in the vicinity of a point.

Horizontal displacement along a trough subsidence is proportional to the slope of the subsidence profile. The horizontal strain at a point is defined by two normal strain components and the shear strain or horizontal distortion. Uniform horizontal displacement seldom causes direct damage to the surface structures. Strains cause distortion, cracking and failure of buildings, pipelines, highways, etc..

Classification of the Mode of Subsidence:

The ratio of the average convergence, S_m, to the depth, H, is the most significant factor (S_m/H) in the prediction of the effects of mining on the surface (Galvin, Stujn and Wagner, 1981; and Fauconnier & Kersten, 1982). Depending on the value of this ratio (S_m/H), mining will affect the ground surface differently if the lateral extent of mining is large enough to allow the full development of subsidence. Salamon (1974) has grouped the surface subsidences into three classes based on the ratio (S_m/H):

1. Discontinuous subsidence: ($S_m/H > 0.1$).

   This type of subsidence is the least predictable and potentially the most hazardous subsidence situation and should be avoided in the presence of structures requiring protection.

2. Subsidence with surface cracks: ($0.1>S_m/H> 0.01$)
   As the ratio value decreases, the subsidence of the surface outside the mined out area becomes more and more clear. Cracks can be seen in zones of horizontal tension. Most structures exposed to subsidence in this group will be damaged.
3. Smooth subsidence: ($S_m/H < 0.0001$)
   The displacement components are continuous functions of the coordinates.

Prevention of Damage to Surface Structures

Underground controlled mining involves techniques that do not produce surface subsidence exceeding the allowable values; it is therefore effective in reducing surface damage. The various methods of protecting surface structures fall into one of the following categories:

1. use of solid supporting pillars,
2. use of some form of pillar mining,
3. packing or stowing,
4. protection by phasing of extraction.

2.4 PILLAR EXTRACTION

There are two practical mining situations in which pillar extraction is required: first, when the mine is either new or new enough to permit a choice of extraction method; second, when it is expedient to extract pillars that have been cut without regard for subsequent recovery. The same principles and techniques of planning for pillar extraction can be applicable to both situations (Galvin, Stujn and Wagner, 1981; and Kresten 1982).

Several factors can affect planning for pillar extraction and are listed below:

1. The necessity to protect surface objects may constrain planning.
2. The relation between production resulting from development and production resulting from pillaring can affect the economics of coal. There may be important differences in cost or quality of coal produced in the two operations.
3. The interval of time between development and pillaring is very important and should be as short as possible.
4. Rock Mechanics and strata control which are major factors in the design of the system.
5. The alternative methods available for extracting pillars are integral with planning.
6. Other features, which, though present in any mining operations, tend to be hazardous in pillar

extraction, such as gas emissions from the caved area, water inrushes, and bumps.

2.4.1 PANEL DESIGN

Mines are typically subdivided into panels which are separated by barrier pillars in order to localize and confine planned or unplanned events such as caving, flooding, explosions, and fires. The design of panel dimensions has been discussed in section 2.3.4. The factors influencing the dimensions of panels are (Mason, 1951; Deshmukh and Deshmukh, 1966; and Stefenko, 1975):

1. Rock Mechanics and Strata Control
2. Security
3. Barrier Pillars
4. Geological Limits
5. Production
6. Ventilation

2.4.2 PILLAR DESIGN

1. Pillar dimensions: - This has been discussed in sections 2.3.1 thru 2.3.3.
2. Mining system: - There are several methods of reducing pillars formed during development. The pillars are designed based on the mining system to be used for retreating.

3. Machinery: - The working dimensions of the mining equipment must be considered when pillar dimensions are determined.
4. Width of roadways: - Productivity in development mining favors a wide room, as opposed to pillaring, in which a narrow room improves roof control.
5. Face length: - The panel or section width is a function of the number of roadways and hence the number of pillars required for maximum productivity of a unit.
6. Sequence of pillar extraction: - To avoid protrusions into the goaf, pillars should be removed in an orderly manner so that the pillar line is always as straight as possible.
7. Shape of pillars: - The shape of the pillars should be suited to the method of mining and may be square or rectangular.

2.4.3 STRATA CONTROL AND SELECTION OF ROOF SUPPORTS:

The abutment high stress zone is unavoidable in total extraction methods and must be carefully controlled. The pillars are designed taking into account this increased stress beyond the normal pillar load caused by mining. High stresses may be expected in (Holland, 1954; Galvin, Stujn, and Wagner, 1981; and Fauconnier, 1982):

1. areas close to the extraction line
2. pillar areas close to wide passageways
3. pillars that are larger than surrounding pillars
4. protrusions on active pillar lines
5. mining seams in which the coal and adjacent strata have very different physical properties.

The position of the peak abutment stress varies from location to location and the zone may extend across a large area. To maintain safe working conditions in the vicinity of the pillar edge, a roof support with high initial stiffness is required. Timber poles are best in this regard.

The most commonly used types of support in total extraction panels are single props such as timber, hydraulic props, roof bolts, and fenders of coal, or a combination of these. Whatever combination chosen must fulfill certain conditions (Galvin, Stujn, and Wagner, 1981; Naismith and Pakalnis, 1982; and Fauconnier, 1982).

1. The method must provide adequate support to the workings during development and allow for the abutment stresses during subsequent pillar extraction.
2. An effective breaker line must be provided to limit the extent of caving, and must be constructed at the goaf edge to protect the working area and to

induce a break in the roof.

3. The speed of erection must match the face advance.
4. The support should not hinder machinery.
5. The purpose of the support is limited to supporting only the immediate roof for as long as is required.
6. Removal must be easy and rapid.

Initial installation cost plays an important role in support selection. Secondary costs such as clean-up of fall material, and lost production due to disabled men and equipment, are often not considered. Proper selection of support type and optimum support capacity can minimize the initial installation and secondary costs while achieving the maximum safety factor. Roofbolts, timbers, and steel rails are commonly used during the development phase of mining and in combination with other roof supports while retreat mining. A greater use of rock bolting in narrower openings, with a minimum of development work, and the application of powered supports to total extraction is the logical approach to reduction of support costs in coal mines (Stefenko, 1975; and Australian Institute, 1976).

Hydraulic mechanized supports are becoming more popular for pillaring operations. However, it has been observed that the hydraulically actuated supports used near a pillaring face require repair before they have reached their rated service life. Hydraulic props get out of order most frequently because the pressure in the hydraulic cylinders at the moment of caving of roof may reach four times the design pressure. Within one shift there are 100 to 150 peaks above the nominal pressure, and this has an adverse effect on the reliability of the hydraulic props and the sections as a whole (Brenner and Lynboshchinsky, 1975).

The distribution of the durations of sudden rooffalls, measured in retreat mining faces, is an exponential distribution [Sadykov and Setkov, 1978] and can be written in the form:

$$F(t) = 1 - e^{-\lambda t}$$

$$\lambda = 1/\bar{t} \qquad (1)$$

and

$$\bar{t} = \sum_{i=1}^{n} t_i/n$$

where n = number of measured rooffall durations, t_i.

To represent the maximum duration of sudden rooffalls, let us take the probability (F_{tmax}) = 0.95. Then from Equation (1) we get $0.95 = 1-e^{-\lambda t_{max}}$, and $t_{max} = -\log(0.5)/\lambda$

Sadykov's report reveals that, like the durations of sudden rooffalls near a retreat mining face, the time between sudden rooffalls is exponentially distributed. The expected value of the time between sudden rooffalls, T_m, can be determined. Knowing the values of T_m we can provisionally calculate the number of sudden rooffalls which must be withstood by a mechanized support during its service life in any particular condition. Obviously,

$$N_1 = \frac{250*t_s*24}{T_m}$$

where N_1 is the number of sudden roof falls, 250 is the number of working days per year for the retreat mining system, t_s is the service life of the system in years, 24 is the number of hours in a day, and T_m is the time between roof falls in hours.

These indices are used during the decision-making process of support type and capacity design. Consideration of the indices, like maximum duration of sudden rooffalls, expected value of the time between sudden rooffalls, and the number of sudden rooffalls which must be withstood by a support system, results in reduction of support costs in coal mines.

The total support cost of each of several alternate ground control systems is accounted for in an effort to determine the most cost-effective roof support for a given mining operation. The total roof support cost model developed by Partrick [1978] is very general and is expressed in terms of gross profit per ton, P_j and the revenue and total costs associated with the j-th support system,

$$P_j = R - G - B_j - A_j - M_j - \frac{F_{tj} V_j}{T_j N_j}$$

where: R is the revenue per ton of coal,

G is the general operating cost per ton of coal,

B_j is the support cost per ton of coal,

A_j is the personnel injury cost per ton of coal,

M_j is the equipment damage cost per ton of coal,

F_{tj} is the total lost production costs,

T_j is the number of tons of coal per shift worked,

V_j is the total volume of rooffalls occurring in the time period analyzed,

N_j is the number of shifts worked.

This model contains every valid element of cost that acts to reduce the per ton profitability of a mining operation, and permits the comparison of alternative roof support systems.

The equation used in the model can be solved deterministically or by using digital simulation techniques. The deterministic solution requires the determination of the subscripted variables from time studies and other sources of historical data such as company cost records. The digital simulation model calculates the various parameters such as equipment damage, the number and extent of injuries, and other rooffall related events based on statistical distributions. These statistical distributions are based on historical data for the support system concerned. The statistical distribution introduces an added degree of realism into the model that is not present in the mean value models.

The profits per ton of coal for room-and-pillar mining operations with and without pillaring operation are evaluated using the total roof support cost model. If retreat mining becomes economically infeasible, then the optimum pillar dimensions can be determined for most profitable percentage extraction by comparing different alternative roof support systems. Hence, the total roof support cost model is an important tool for the economic feasibility analysis of retreat mining.

2.4.4 METHODS OF PILLAR EXTRACTION:

Pillar extraction or retreat mining techniques vary widely over the United States. The widely-used processes are split-and-fender, pocket-and-wing, outside lifts, and open ending. There are more techniques of pillar extraction which are commonly used overseas. A brief discussion of all processes is presented here.

2.4.4.1 Split-and-Fender

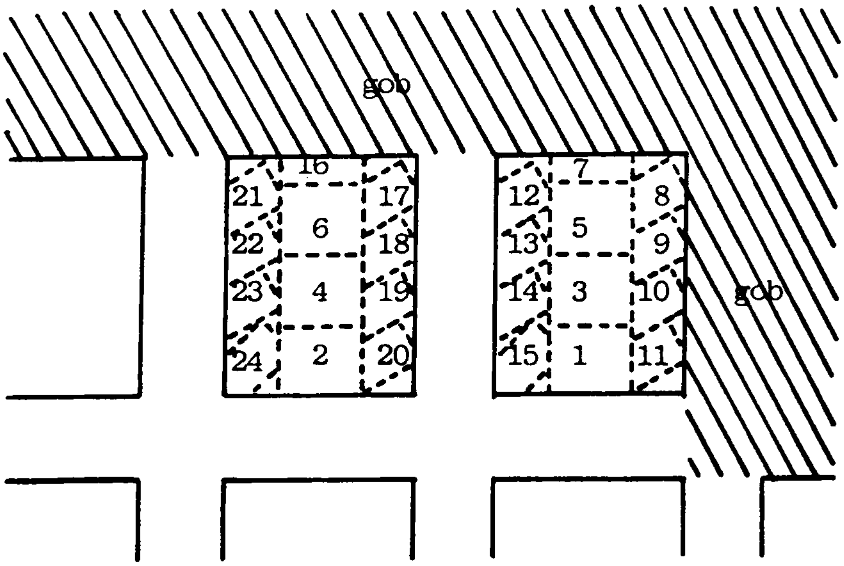
This method is used for extracting relatively small pillars but a large pillar can be extracted using multiple splits. A sequence of cuts is mined through the pillar parallel to the pillar's long side, and a split is formed with the creation of two fenders of coal. The roof within the split is supported by bolts, posts and timbers or a combination of these and the fenders are extracted from the split or adjacent entry with additional support provided by posts. More than one pillar is extracted at the same time and a typical sequence of cuts is shown in Figure 2.16. The numbers represent the cut sequence of the two pillars for continuous mining equipment.

2.4.4.2 Split-and-Fender Modifications

As split-and-fender is the most common pillar extraction method in the United States, it is also the most widely modified (Kauffman and Hawkins, 1981). Six of the variations are very common and will be discussed here.

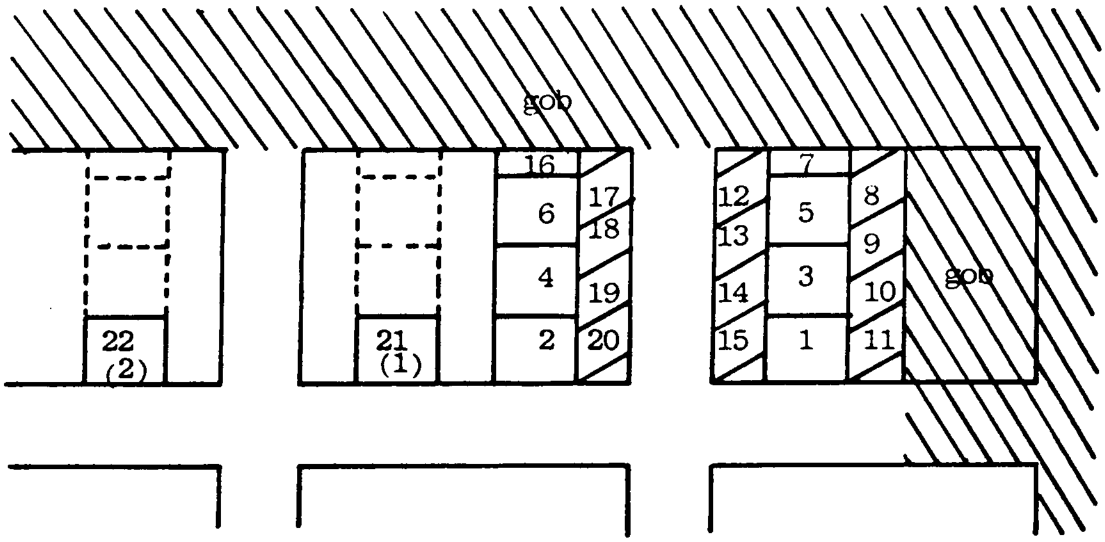
Multiple Splits: If the pillar width is more than 45 feet, one split is not sufficient to extract the remaining fenders. For larger pillars, many mines use two to three splits. Figure 2.17 shows a typical cut sequence for extracting pillars using two splits. The pocket-and-wing extraction process may be preferable to the use of a multiple-split-and-fender process for extracting large pillars.

Christmas Treeing: Two fenders are extracted from the same split or entry. There are two ways to do this. The first method is depicted in Figure 2.18, and involves the extraction of a single pillar. Cuts are made to the left



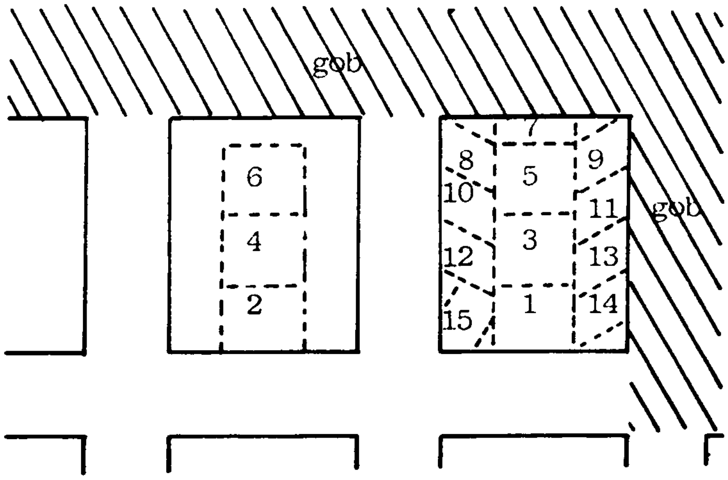
Split-and-fender cutting sequence.

Figure 2.16



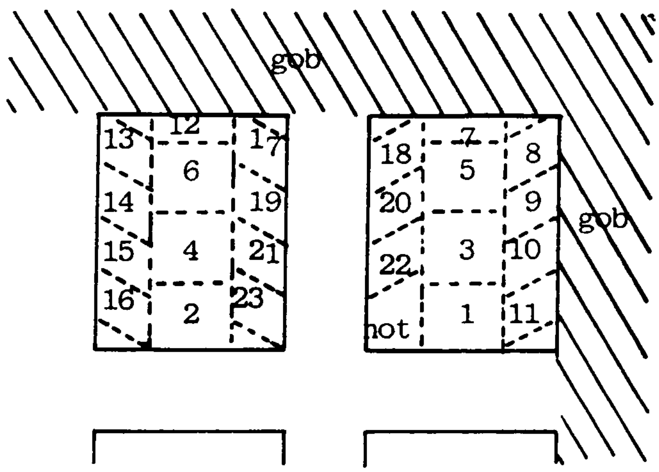
Multiple Splits, (After Kauffman)

Figure 2.17



Christmas-treeing cutting sequence (one pillar).

Figure 2.18



Christmas-treeing cutting sequence (two pillars).

Figure 2.19

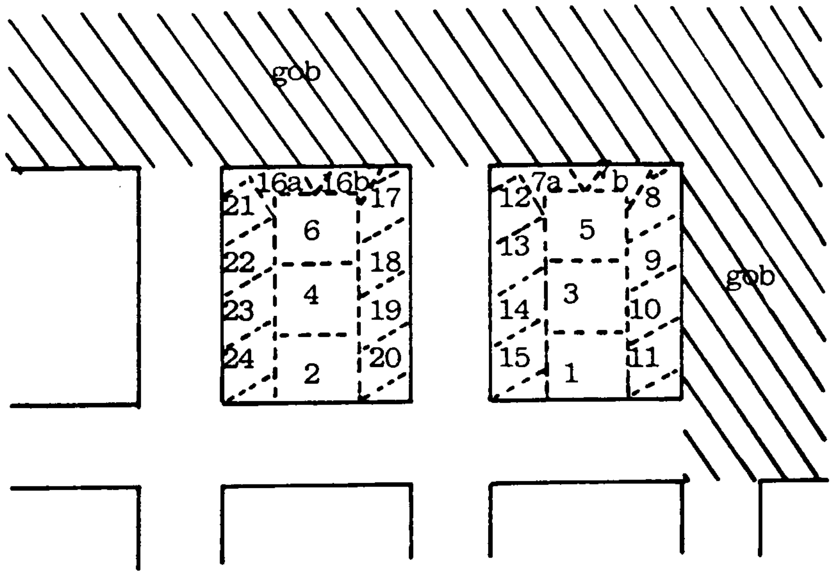
and then to the right. The second method, depicted in Figure 2.19, extracts two pillars from the same entry. Christmas treeing methods are not used under normal situations because of the large unsupported roof span over the continuous miner.

Indicator Stump: The final cuts in the split are angled outward in such a fashion that a wedge-shaped stump is left in the center of the end of the split. The stump is used as an indicator of roof activity while the inby fenders are being extracted. The cut sequence is shown in Figure 2.20.

Fender Breakthrough: - When fenders are less than 8 feet thick, this technique is used to extract pillars. The miner breaks through the inby fender into the gob as the split is being advanced. These breaks aid in ventilation and the thickness of the fender can be indicated. A cut sequence is shown in Figure 2.21.

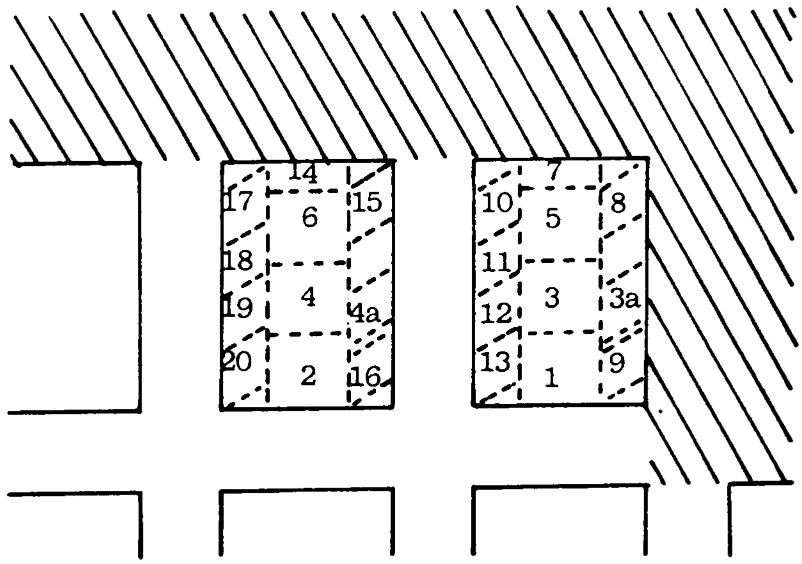
Fender Notching: - In this method, a notch is cut to the left of the split as the fender to the right is extracted. The positioning of the notch cuts coincides with the lifts that will be removed from the outby fender when that fender is recovered from the entry or split. The cut sequence is shown in Figure 2.22. Because the notch cuts are taken from under supported area, no hazardous exposure is created.

Conventional Split-and-Fender: - When hard coal or coal



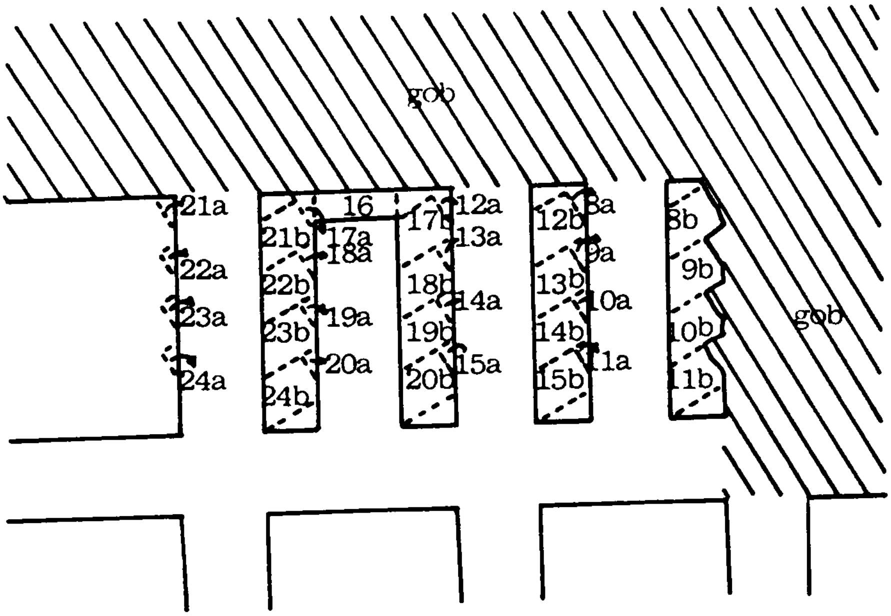
Indicator-stump cutting sequence, (After Kauffman),

Figure 2.20



Fender-breakthrough cutting sequence,(After Kauffman)

Figure 2.21



Fender notching cutting sequence

Figure 2.22

layered with rock is mined, other methods of pillar extraction are not efficient. The conventional split-and-fender technique increases the recovery rate because it confines the coal between the fenders during heavy blasting. An extraction sequence is depicted in Figure 2.23.

2.4.4.3 Pocket-and-Wing

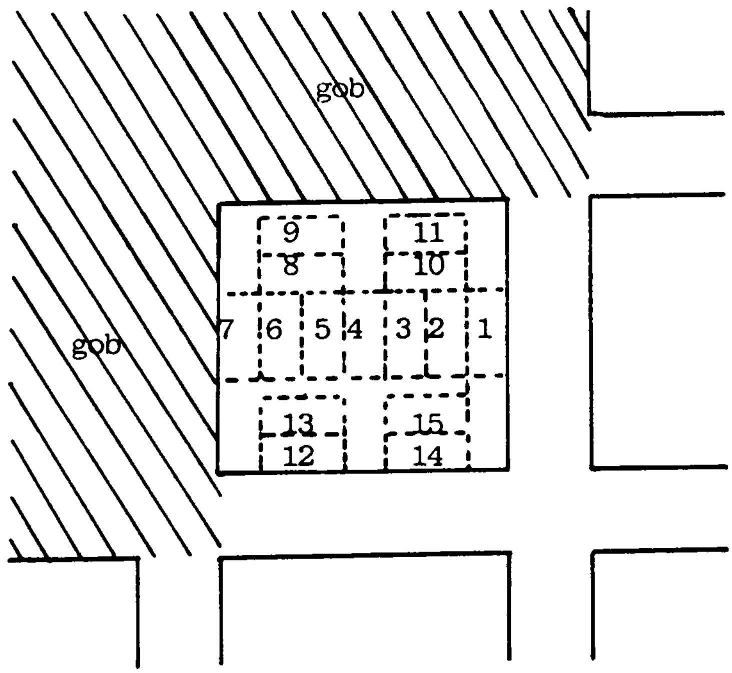
This method is used to extract large pillars and allows two working places within the same pillar. Pockets are formed on the gob sides of the pillar, and lifts are mined between pockets. A fender of coal is left between the pocket and the gob. The wing is extracted after the completion of pockets. More pockets are driven and wings extracted until the pillar is reduced to a final stump. This stump is recovered from the intersection. The sequence of cuts is depicted in Figure 2.24.

2.4.4.4 Nonsequential Pocket-and-Wing

This is a variation of the pocket-and-wing method. It is made possible by the use of continuous miners with a bolting function or the use of boring type continuous miners. It is used in the areas of high gas and weak roof, since air can be directed to a single face throughout the extraction period. A cut sequence is shown in Figure 2.25.

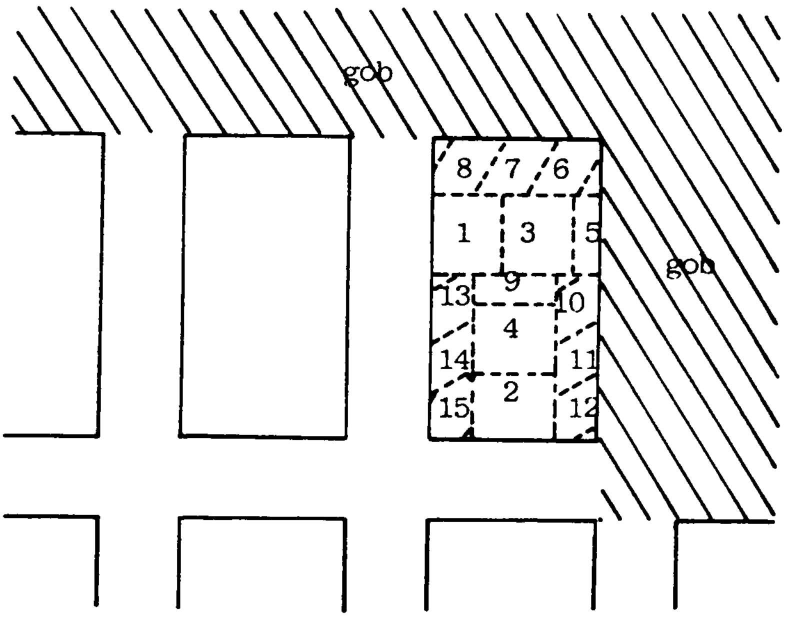
2.4.4.5 Outside Lifts

This method is very suitable for small pillars. The



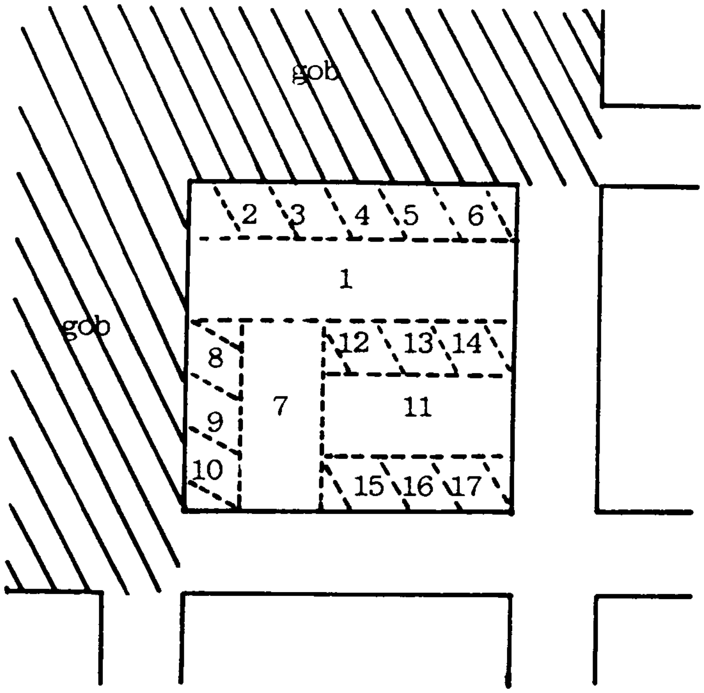
Split-and-fender cutting sequence using conventional -mining equipment, (After Kauffman,1981)

Figure 2.23



Pocket-and-wing cutting sequence

Figure 2.24



Nonsequential-pocket-and-wing cutting sequence

Figure 2.25

pillar is dimensioned in such a way that lifts can be taken from one side of the pillar without going beyond the supported roof. Lifts are taken beginning near the gob and moving toward solid coal. The sequence of cuts is shown in Figure 2.26.

2.4.4.6 Outside-Lift Variation

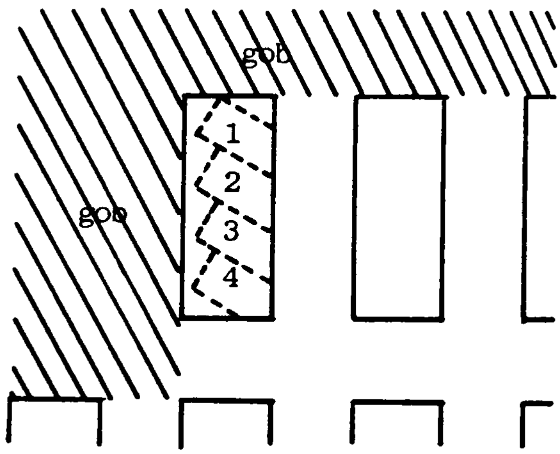
A combination of outside lifts and Christmas treeing, the outside-lift variation can be used to extract pillars up to 30 to 35 feet wide and at a low overburden pressure. This process does not require roof bolting. A sequence of cuts is depicted in Figure 2.27.

2.4.4.7 Open Ending

In the open-end method a lift is mined off one side of a block adjacent to a previously mined and caved area, or gob. The width of each lift is determined by physical conditions and the type of equipment used. The individual cuts of coal in a lift are mined out completely to the gob, creating an open side or end. The roof over the cut is supported. This process is widely used in conventional mining sections. Cuts can be taken from as many as six pillars simultaneously. The pillar line is generally at a 45 degree angle. The sequence of cuts for one pillar only is shown in Figure 2.28.

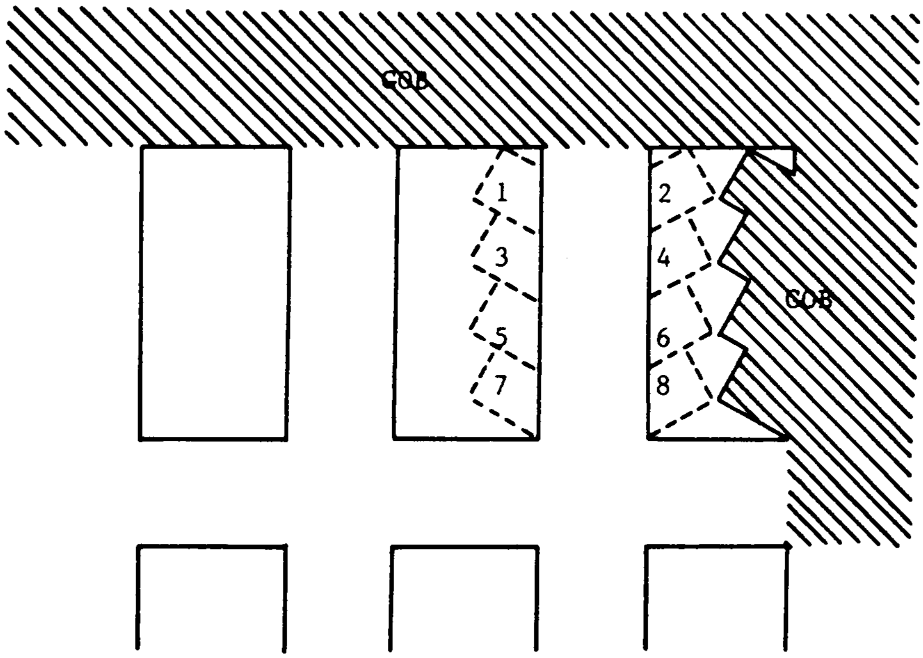
2.4.4.8 LONGWALL CAVING ON KNIFE EDGES

This method of pillar extraction has been practiced



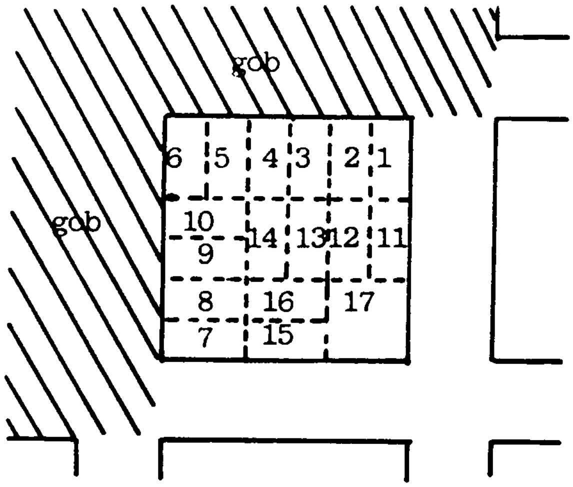
Outside-lift cutting sequence, (After Kauffman)

Figure 2.26



Outside-Lift Christmas Treeing cutting sequence

Figure 2.27



Open-ending cutting sequence

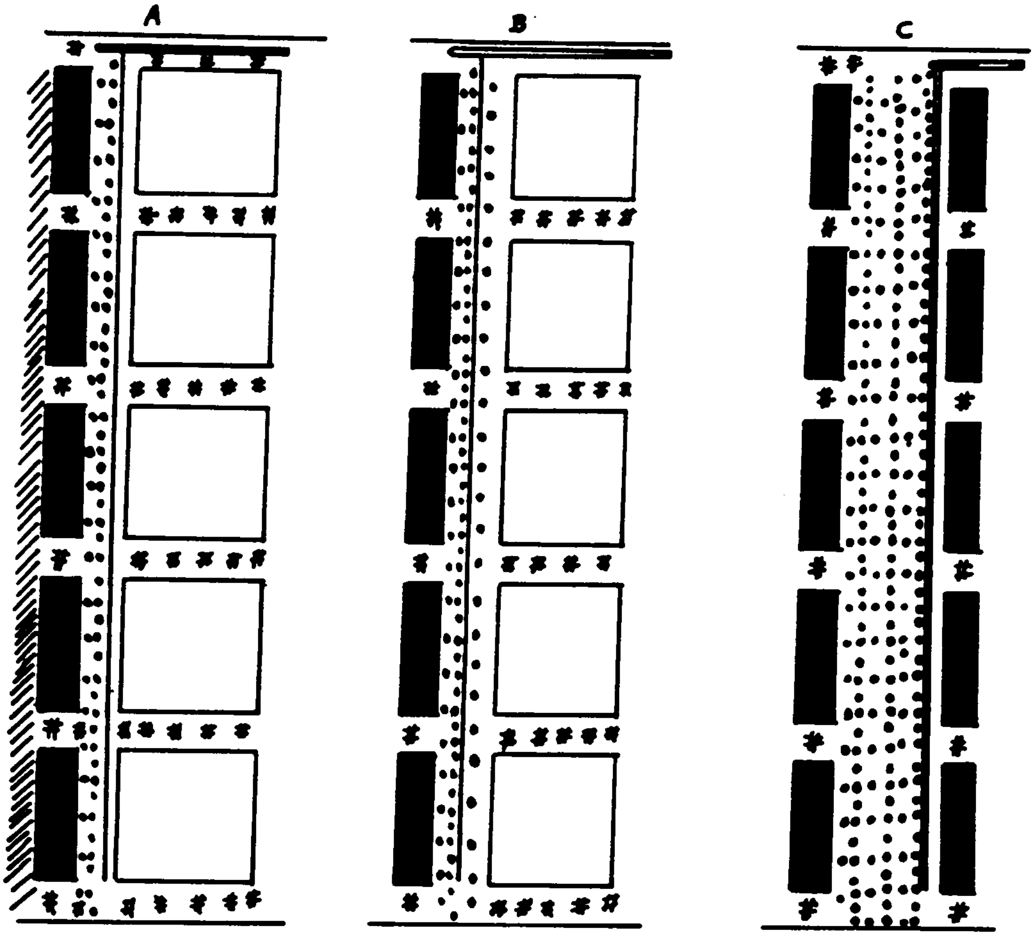
Figure 2.28

abroad successfully and consists of cutting a row of pillars by longwall cutting and flight loading machines. Two machines are used simultaneously. Both machines start working in opposite directions from the central pillar of the panel and undercut the pillars in both directions (Mrig, 1970). While cutting, wooden wedges are inserted in the cuts for supports. After cuts are made in all the pillars of a row, the pillars are blasted one by one. Then both machines start moving towards each other, loading the blasted coal. The coal cutting machines are fitted with chain and flights to load the coal into the conveyor installed along the face. The conveyor takes this coal to the end of the panel (row) and discharges to the chain conveyor installed in the end entry. When the loading is going on, one gang of timber men work with each machine to erect a row of supports behind the machine, leaving a clearance of 5 feet between the face and the support. After loading the blasted coal from the whole face, both the machines are again brought to the cutting position and the conveyors dismantled and re-erected along the face, leaving a clearance of 4 feet for the machines to move. The machines again move in opposite directions, under-cutting the pillars, and the process is repeated. The conveyor is dismantled after every cut and re-erected along the face. When the pillar rib width becomes 13 feet, the whole conveyor is dismantled and erected in the next gallery. The machines are also shifted and the gate chain conveyor is also cut-off to that extent. The timber supports set in the extracted area are then withdrawn one by one and as many props as possible are also withdrawn. The 13 feet pillar ribs help in bringing the roof down in the gob, working as knife edges and preventing the overriding of gob into the working area. The layout of the pillar extraction section is shown in Figure 2.29. The main supports used in the area are rows of props and rows of cribs alternatively set along the line joining two ends of the panel.

In the first row, after the withdrawal of the supports, the roof may be blasted down if it does not come down easily. Experience shows that from the third row onwards, the roof comes down itself, as the supports are withdrawn even for strong roof (Mrig, 1970).

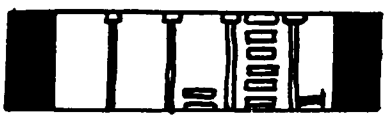
2.4.4.9 MODIFIED LONGWALL METHOD

This method of pillar extraction is similar to the previously described method 'Longwall Caving on Knife Edges' but the 4 foot wide pillar ribs are also extracted from the next gallery. The existing galleries during mining by the longwall method are crossed by leaving 4 foot wide pillar ribs and shifting the conveyor to the new position in the gallery in front of the steel props. Supports behind the ribs will be withdrawn and shifted to the new gallery.



A First Cut being given B After first cut has been loaded

C



Face at the time of last cut

Longwall Caving on Knife Edges, (After Mrig)

Figure 2.29

The roof behind the rib is brought down by induced caving if necessary before starting extraction of the rib. While extracting the rib, one row of supports should be erected with breaker props. After the rib has been extracted the last line of support adjacent to the gob side will be withdrawn and the roof made to cave (Skowron and Burszczyk, 1976).

2.4.4.10 PILLAR SPLITTING

Partial extraction is achieved using the pillar splitting method of pillar extraction. Each pillar is split into four stooks by driving one gallery along the length of the panel and one gallery along the width of the panel. The width of the split gallery is the same as the width of the main gallery. The galleries and splits are supported by steel channels or strong timber supports.

In the past, the splitting of pillars as a final operation was practiced in some mines without many technical considerations. As a result, after several years, there have been cases of sudden surface collapses. Hence, the need exists to scientifically study and determine the extent to which the pillars can be reduced without affecting the surface while giving the maximum percentage of recovery (Ali, 1971).

3. ESTIMATING PILLAR EXTRACTION COST

In the formulation of the model, mining cost components are classified either as capital or operating costs. Pillar extraction does not require any additional capital investment to operate the mine since supply, haulage, ventilation, and power systems were established during the development phase of mining. Operating costs are the major costs during pillaring.

Operating or pillar extraction costs may be classified as direct, indirect, fixed, and subsidence compensation costs.

Direct costs are those costs which tend to be proportional or partially proportional to production, such as the cost of operating supplies, utilities, direct operating labor, operating supervision, maintenance supervision, payroll overhead, and union welfare (Humphreys and Katell, 1981). Direct costs include both variable costs and semivariable costs. Variable costs are those which tend to be a direct function of mine production or output, while semivariable costs are direct costs which are only partially dependent upon production or output. Semivariable costs, which include direct labor, supervision, maintenance, general expense, and overhead costs, increase with production rate but not in direct proportion. When the mine is not operating, a portion of the semivariable costs continues to be incurred. At zero production, semivariable costs generally total about 20 to 40 percent of the total semivariable cost at full production (Katell, 1981).

Indirect costs include plant overhead (administration, laboratory, shops and repair facilities) (Humphreys and Katell, 1981).

Fixed costs which are independent of mine production and are incurred whether or not the production rates change, include property taxes, insurance, and depreciation.

Subsidence compensation costs are the full compensation in money for subsidence damages or control costs.

Production cost estimates can be performed on a daily, unit-of-production, or annual basis. Humphreys and Katell (1981) recommended an annual computation for the following reasons:

--- It "damps out" seasonal variations.

--- It considers equipment operating time.

--- It is readily adapted to less-than-full capacity operation.

--- It readily includes the effect of periodic large costs (scheduled maintenance, vacation shutdowns, etc.).

--- It is directly usable in profitability analysis.

--- It is readily convertible to the other bases, daily cost and unit-of-production, yielding mean annual figures rather than a potentially high or low figure for an arbitrarily selected time of year.

3.1 DIRECT COSTS

Operating Supplies (rockbolts and timbers, mining machine parts, lubrication and hydraulic oil, rock dust, ventilation, bits, cables, etc.), Utility, Royalty, Payroll Overhead, Union Welfare, Reclamation Fund, and Licenses costs for use in this study are described below.

3.1.1 Labor, Supervision, and Maintenance Costs

Labor costs are the wages and salaries paid to labor for the operation and maintenance of equipment and mine facilities. It is a dominant factor in an operating cost estimate and depends on the number of individuals required for production, the wage rate required to secure appropriate labor, and hours worked. To properly estimate this cost, a 'manning table' must be established in as detailed a manner as possible. Katell (1978) suggested that this table should indicate the following:

--- The particular skill or craft required in each operation

--- Labor rates for the various types of operations

--- Supervision required for each process step

--- Overhead personnel required

These factors should be included in the manning table for maximum estimate accuracy. A typical manning table for an underground coal mine production section as estimated by Katell (1974, 1975, 1978) is shown in Appendix G. Once the manning table is developed (at a minimum including all direct production labor), labor costs can readily be estimated from company records of wages and salaries by position, union wage scales, salary surveys of various crafts and professions, or other published sources. Because labor costs are prone to rapid inflation, often at rates sharply different from general inflation rates, care must be taken to obtain current figures and to properly project future wage rates (Haskins, 1978; Humphreys and Katell, 1981)

Top management salaries are considered supervision costs and should be established through a manning table as discussed above.

Maintenance costs are a semivariable category, generally distributed about 60% to labor and 40% to materials. Maintenance generally increases with the age of equipment. Maintenance labor costs, like supervision costs, should be delineated in the labor manning table (Humphreys and Katell, 1981).

3.1.2 Operating Supplies Cost

During the pillar extraction process, supplies are required to fuel and lubricate equipment and machinery, support roofs,replace defective or worn out parts, etc. Operating supplies also include explosives and other materials that may be required under certain mining conditions. Rockbolts and timbers are important operating supplies during the pillar extraction process.

3.1.3 Power Cost

Mining operations require significant amounts of electric power to operate mining equipment and facilities. Power costs decrease as demand decreases in the market. Electric power charges are usually based upon two factors: (a) demand factor -- the maximum power [kW] per month, (b) energy [kW-hr], (Humphreys and Katell, 1981; Kolstad, 1974).

3.1.4 Water Cost

Water is used in underground mining operations for dust control, bit cooling, etc. The requirement for water is determined including a reasonable allowance for sanitary water, etc., and water cost is estimated. The water cost decreases as demand for water decreases.

3.1.5 Royalty Cost

The royalty cost is the expense of obtaining sufficient coal reserve leases for an appropriate length of time to enable production (Haskins, 1978).

Steele (1964, 1967) defined the royalty:

> "A royalty is not a rent,though often is called. For, except when mines, quarries etc. are practically inexhaustible, the excess of their income over their direct outgoings has to be regarded, in part at least, as the price got by the sale of stored-up goods -- stored up by nature, indeed, but now treated as private property,......"

Royalties may be variable, semivariable, or fixed depending upon the conditions of the royalty agreement. Payments in proportion to production or fixed payments per annum are treated as direct operating costs (Humphreys and Katell, 1981).

The royalty payment made to the owner of mineral rights complies with a long-term contractual agreement authorizing the coal company to extract the coal, usually until the exhaustion of the coal subject to the contract. The coal mining company that can acquire its coal under a long-term contract and is only required to make payments on the resource that has been mined obviously has more funds for investing in machinery, equipment, and other capital requirements than the company that must purchase total reserve requirements before mine development (Haskins, 1978; Leisenring, 1973).

3.1.6 Payroll Overhead Cost

The payroll overhead cost is an expense, exclusive of wages and salaries, that must be paid by the coal company. This cost, associated with employee "fringe benefits," includes workers' compensation, pensions, group insurance, paid vacations and holidays, social security, unemployment taxes and benefits, and profit sharing programs. The extent of these costs varies from industry to industry and company records are the best measure of their magnitude. Coal companies in the United States estimate payroll overheads at 40% of direct labor plus supervision plus maintenance labor costs. For other countries this factor is adjusted to suit local conditions (Humphreys and Katell, 1981).

3.1.7 Union Welfare Cost

This cost is also called the union welfare contribution. Haskins (1978) discussed the union welfare cost as a monthly royalty payment on each ton of coal produced for use or sale by mine operators. Employees may receive a wide range of additional benefits and services, described as "fringe benefits," which are not considered part of wages and salaries (Heneman, 1965; and Haskins, 1978). This is true of employees who work for coal companies signatory to the National Bituminous Coal Wage Agreement. Through the collective bargaining efforts of the United Mine Workers of America, Welfare and Retirement Fund fringe benefits such as pensions, widow and supervisor benefits, hospital and medical care benefits are established (Haskins, 1978; and Subcommittee on Labor, 1970).

3.2 INDIRECT COST

According to Haskins (1978), and Katell (1978), indirect cost is an expense item added to cover those costs arising from the use of labor and supplies that are not individually itemized under other variable costs. The most common way of computing indirect costs is to take a percentage of labor, supervision, and operating supplies. The percentage figure for computing indirect costs in pillar extraction systems is 15.

3.3 FIXED COSTS

Fixed costs are costs that do not vary with the level of output (Haskins, 1978). These costs are incurred whether or not production rates change. Pillar extraction or retreat mining fixed cost components include taxes and insurance, and depreciation.

3.3.1 Taxes and Insurance

Insurance is an expense for the risk of loss due to unknown hazards. Insurance covers such items as damage or destruction of mine facilities and personal or corporate liability. Insurance and mine property taxes must be included in the production cost estimate. In coal mining areas these costs total 2 percent of mine costs per year (Haskins, 1978).

3.3.2 Depreciation Cost

According to Whitney (1979), depreciation is the periodic write-off of the cost of items of property and certain other long-lived tangible assets. It denotes a periodic cost allocation against revenue of tangible assets such as buildings, machinery and equipment. Depreciation is also the process by which the capitalized cost of a fixed asset is recovered over its estimated useful life.

The purpose of depreciation is to allow credit against operating costs, and hence taxes, for the non- recoverable capital expense of an investment.

There are four depreciation methods applicable to any kind of depreciable asset, which are listed below:

(i) Straight-Line Depreciation

(ii) Declining Balance Depreciation

(iii) Unit-of-Production Depreciation

(iv) Sum-of-the-Years-Digits Depreciation.

Straight-Line Depreciation: This is an often-used method of computing depreciation. To calculate straight-line depreciation, annual depreciation is simply made equal to the depreciable portion of the initial capital investment divided by the IRS-approved depreciable life of the project. Mathematically:

$$D_{sl} = C/L$$

where

D_{sl} = annual straight-line depreciation

C = depreciable portion of the capital investment

L = IRS - approved life, in years.

Declining Balance Depreciation: This method provides for annual depreciation charges which cannot exceed 150% of the applicable straight-line method (Whitney, 1979). The depreciation allowance for year n in the life of the asset is calculated as follows: First subract the accumulated depreciation to date from the cost basis of the asset; multiply this difference by 1.5, and divide the result by the life in years of the asset. Mathematically:

$$Dn = (CB - \sum_{i=0}^{n-1} Di)(1.5/L)$$

where

Dn = depreciation in year n

CB = cost basis of the asset

$\sum_{i=0}^{n-1}$ Di = accumulated depreciation to date

L = life of asset in years

However, since the item can not be depreciated below its salvage value, the difference between CB and the accumulated depreciation to date plus salvage value is compated with Dn.

If $Dn > (CB - \sum_{i=0}^{n-1} Di - \text{Salvage Value})$

then

$$Dn = (CB - \sum_{i=0}^{n-1} Di - \text{Salvage Value}).$$

Thus, we do not depreciate the item below its expected salvage value.

<u>Unit-of-Production Depreciation</u>: The annual deduction under this method is determined by dividing the unrecovered basis (less salvage value) of the asset by the number of units the asset is expected to produce during its remaining useful life, and multiplying the result by the number of units produced during the tax

year (Whitney and Whitney, 1979). Mathematically:

$$Dn = (CB - \sum_{i=0}^{n-1} Di - SV)\ (Pn/Pr)$$

where

Dn = depreciation for year n

CB = cost basis of the asset

$\sum_{i=0}^{n-1} Di$ = accumulated depreciation to date

SV = salvage value

Pn = production during tax year

Pr = production remaining at the begining of year.

There are many other acceptable depreciation methods, of which the double-declining balance and sum-of-years-digits methods, accelerated depreciation, are the most commonly used (Humphreys and Katell, 1981).

Sum-of-the-Years-Digits Depreciation: In this method, a changing fraction is applied to the original cost of the asset (less salvage value). The denominator of this fraction always remains the same and is the sum of the numbers representing all the years in the life of the asset. The numerator is the number of years remaining in the useful life of the mine property (Whitney and

Whitney, 1979). Mathematically:

$$Dy = C\ (2(n-y+1)/n(n+1))$$

where

Dy = depreciation in year y

C = depreciable portion of investment

n = asset life, in years.

3.4 DEPLETION ALLOWANCE

The depletion allowance is a charge against income for the usage of non-renewable natural resources. There are two methods of computing depletion allowance. They are cost depletion and percentage depletion. The method which results in the greater depletion allowance for any taxable year is selected for calculating depletion allowance.

3.4.1 Cost Depletion:

Cost depletion is based upon the cost of mineral property, the number of units of mineral sold during the year, and the number of units of mineral remaining in the deposit at the end of the year (Whitney and Whitney, 1979). Mathematically:

$$CDn = (CB - \sum_{i=0}^{n-1} Di)\ (Un/(Un+Ur))$$

where

CDn = cost depletion allowance in year n

CB = cost basis of the property

$\sum_{i=0}^{n-1} Di$ = accumulated depletion taken in preceeding years (both cost and percentage)

Un = units of mineral sold during the year n

Ur = units of mineral remaining at year end.

3.4.2 Percentage Depletion:

This is an allowance expressed as a percentage of the gross income from the mine and varies in amount with the type of mineral. It is limited to 50 percent of the taxable income from the mine before the depletion allowance is deducted. Mathematically:

$$PDn = (MPDA)\ (GIMn - RPn)$$

If $PDn > (0.5)(TIBD)$

then $PDn = (0.5)(TIBD)$

where

PDn = allowable percentage depletion for the mineral sold during year

$MPDA$ = mineral's percentage depletion allowance

$GIMn$ = gross income from mineral sales in year n

RPn = royalty payments

$TIBD$ = taxable income before depletion.

3.5 SUBSIDENCE COMPENSATION COST

Subsidence compensation or mining damage cost is one of the important factors governing the selection of a mining method. If the coal seam lies under a densely populated urban area, total extraction of coal becomes economically infeasible. Coal pillars would have to be left or the open area underground would have to be backfilled to prevent surface subsidence, resulting in higher subsidence contol cost.

According to federal government estimates (Bruhn, et al. 1982), underground coal mining causes $25-35 million in subsidence damage to residences, schools and commercial/industrial buildings and $3-4 million in damage to roads, utilities and services in the United States each year. By the end of the 20th century these types of subsidence damage may total more than $1 billion (Comptroller General, 1979). Based on the 1981 dollar, repair costs --the costs to repair major functional and structural damage -- range from a few hundred dollars to more than $100,000, the median being $6,000 to $10,000, per home (Figure 3.1). The repair cost is generally less than 20 percent of the replacement value of the home, and is commonly about 10 percent (Figure 3.2). Surface subsidence may result in excessive structural damage particularly in highly developed urban areas and may reduce land values.

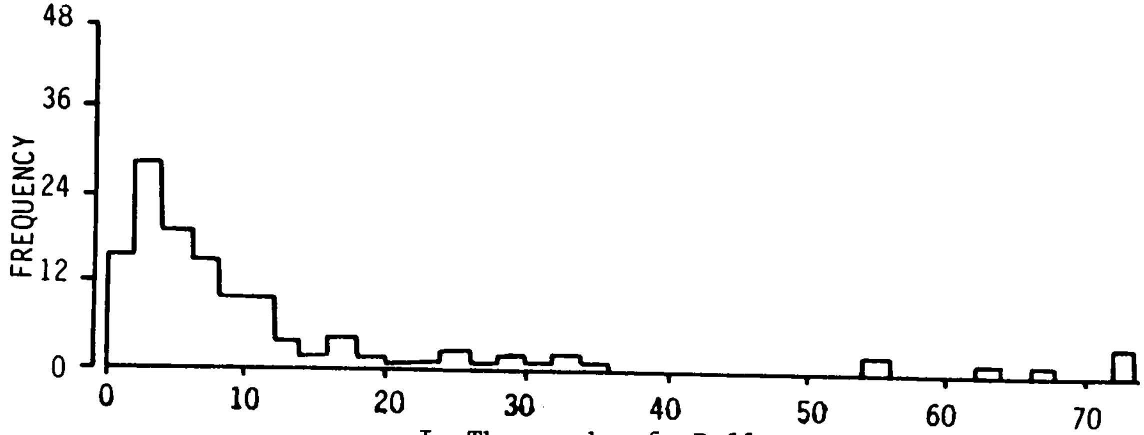


Fig. 3.1 Repair Costs -- Mine Subsidence Damage to Homes in Western Pennsylvania

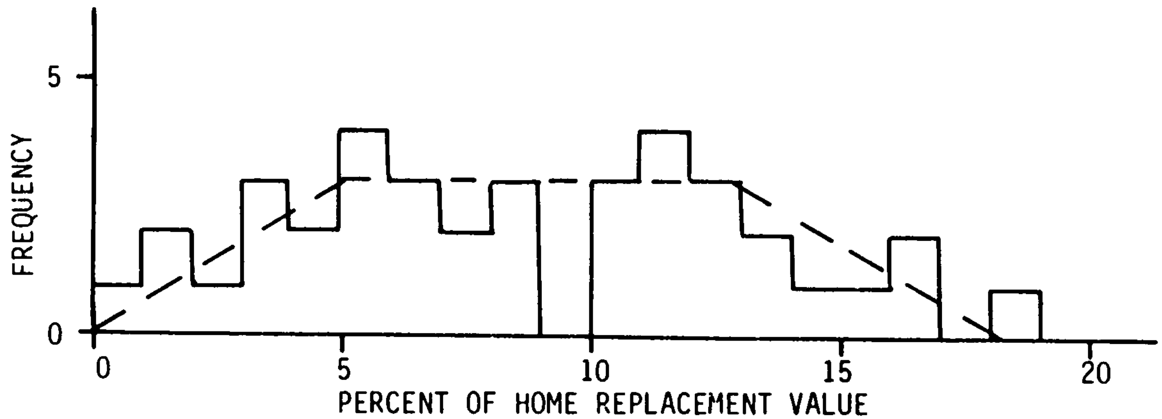


Fig. 3.2 Total Estimated Repair Cost as a Percent of Home Replacement Value

Subsidence losses and control costs are absorbed by the mineral industry as a production cost. These costs are dependent on many factors such as local geologic conditions, land surface utilization, and mining procedures, all of which vary from one area to another (Cochran, 1971).

According to Kratzsch (1983), complete compensation for subsidence damage means restoration to the earlier, economically usable state (natural restoration) or - especially with irreparable permanent damage and with prejudice to asset values because of the expectation of future subsidence - full compensation in money. Subsidence compensation also extends to appurtenances, that is, those things and those rights which serve the economic aim of the main item (service mains, wells, losses of rent, etc.) or are associated with it as movable objects (seed-corn, railway rolling stock, farm tractors, etc.).

There are five different practical mining situations in which surface subsidence may occur and these situations are listed below:

--- mining under a barren area

--- mining under the buildings

--- mining under agricultural land

--- mining under an area which has communications installations

--- mining under areas which have a mixed combination of all the situations listed above.

Pillar extraction or retreat mining under a barren area does not require any subsidence compensation cost. Barren areas can be left without using damage control measures; therefore, there will be no control cost involved in these areas.

Retreat mining under an urban area or buildings causes tilt in buildings through uneven subsidence. When a house becomes tilted its value is reduced due to following reasons (Kratzsch, 1983):

--- difficulties in use

--- unusual appearance

--- loosened framework

--- impaired resale possibilities

--- diminished rentability

--- curtailed returns.

German engineers (Kratzsch, 1983) formulated in 1946 the relationship between tilt and reduction in value. The reduction in value increases progressively as a percentage of the building value, up to a reduction of seventy five percent at a tilt of 50 mm/metre length of the building (Leyendecker, 1946). An extra 25% is added for loosening of structural bonds and there is a linear reduction in value of 1% for each 2 mm/metre of tilt (Vennhofen, 1983).

A new basis which is called the agreed basis and is the result of an agreement between the mining company and the association of property owners is commonly used to compute reduction in value of building (Kratzsch, 1983). According to the agreed basis, a rate of 2% for each 2mm/metre tilt for tilts of over 20mm/metre is used. The agreed basis reduction in value also includes structural loosening and other losses in value.

The value of a building in the event of mining subsidence damage is determined by considering the normal construction cost of an equivalent replacement building at 100% of the 1913 price basis (Kratzsch, 1983). An inflation supplement based on the official building price index and an age depreciation allowance corresponding to the time the building has stood are used to adjust the 1913 reference price value for the building to give the time-related value on the day of damage. Mathematically (Kratzsch, 1983):

Value of Building = (Construction cost 1913 X Building-cost Index /100) - (Age-depreciation Allowance)

The Building-cost index is used as a multiplier to adjust the construction cost for the year 1913 to the present-day price level (new-building value).

The drop in value of old houses is assessed using 1%

deduction per year.

Another prevalent problem caused by mine subsidence is damage to farm land. Subsidence causes an influx of ground water into mine workings, which results in a depletion of farmers' boreholes in the surrounding areas. Ground water becomes useless for farming because of this underground contamination. In addition, the farm land can be displaced horizontally, along with neighboring pieces of land, by a metre or more because of underground mining. Reduction in the size of farm land in the compression zone overlying the extraction area can mean a reduction in assets (Kratzsch, 1983).

The farm land may be classified according to its use as cropland, with the highest value; grazing land, with a lesser value; and forest and unused land, with the least value (Cochran,1971). Compensation for vegetation damaged by mining-induced changes in the ground-water level is calculated taking into account normal yields and the percentage loss in produce. Damage to growth and differences in yield are estimated by field inspection or by weighing the produce and comparing with normal yields.

Mining is usually not conducted in industrial-commercial property areas or areas having communications installations. The cost of measures taken by the mining company directly and solely for the security of a communications installation must be estimated. Coal left in the ground for the purpose of controlling surface damage to communications installations is a subsidence cost and the in-ground value of coal is estimated.

Backfilling of mine voids as a phase of active mining to control surface subsidence is a common procedure in some foreign countries with limited coal resources. The cost of backfilling or stowing is estimated to compute subsidence damage control cost.

Special materials and methods of construction, such as rigid foundations or flexible structures, are used in some mining areas as a means of controlling future subsidence damage. The costs of these surface constructions is estimated to compute control cost (Cochran, 1971).

4. DEVELOPMENT OF MODEL

To more fully develop the potential of room and pillar mining a pillar extraction cost model and computer program was developed to facilitate cost and production optimization. The model provides cost estimates based on the site specific, geologic, and mining environmental factors.

The main computer program was named PILCOST and written in FORTRAN-77. Since the computer program was developed to run on microcomputers, it was divided into different subroutines due to the memory limitation of the compiler (64 K-Byte). The program is simple to run using any FORTRAN-77 compiler developed for microcomputers. It is user-friendly, and input data to the program can be prepared easily based on field observations and case studies. The algorithms used in the program were based on pillar design and operating cost equations.

The model does not consider pillar extraction requiring any additional capital investment since supply, haulage, ventilation and power systems were established during the development of mine. It was developed to analyze two mining situations: 1) a virgin property of sufficient extent to permit pillar extraction, and 2) pillars left previously without regard to pillar extraction. In the first situation, the model can compute the safe optimum pillar dimension for mine development in conjunction with pillaring costs. In the second situation, the model will compute the cost of pillaring using the existing pillar dimensions.

The relationship between pillar dimensions and pillaring methods was specified in the model. The extended life of the mine due to pillaring was based on the ratio of pillars extracted to pillars developed during a given time, and has been considered in the model. An initial situation of a coal seam five feet thick was chosen as a basis for model development.

Sixteen basic assumptions were made while developing the cost model:

- All the coal pillars in a panel are of uniform size and shape.
- All the barrier pillars are of uniform size and shape.
- The number of coal pillars in a panel is the same for each panel; consequently, all panels have equal width and length.
- The overburden load is constant over a coal pillar.
- An element of ground at a depth is subjected to vertical load only.

- The load is uniformly distributed over the cross-sectional area of the pillar.
- The roof strength has no effect on the pillar strength.
- The floor has no effect on the strength of pillar.
- Timber props and rockbolts are the main roof supports used in a pillar extraction panel.
- The ratio of pillars extracted to pillars developed is constant for each panel during the pillaring period.
- The number of roofbolts and timber posts required in splits is fixed for split widths greater than 16 feet and less than or equal to 20 feet (this is due to federal law restriction which permits a maximum distance of 4 feet between two roofbolts).
- Coal reserve is based on 1,800 tons of coal per acre-foot (weight of one foot cube of coal times number of square feet in one acre).
- An average value of seam thickness is used to compute total coal reserve.
- The cost data available from coal mines and USBM reports will be applicable to similar mines of the same capacity.
- The average value of cost adjustment factors are used (specific cases may be different from the average).

- There is no subsidence compensation involved in a pillaring operation under a barren area.

Some special problems were also encountered in the development of the model. Because very few pillaring operations cost data were available, there was a problem of estimating the cost data for any mine size or capacity. With limited cost data it was observed that the cost of mining was not linearly dependent on the size or annual production capacity of the mine. The unknown cost of a mine of any desired size was related to the known cost of a mine of specific size by using the cost capacity factor and interpolation method.

Several coal pillar design formulae are available and all of them have been applied extensively in different coal regions. There was a problem of choosing the most suitable formula to incorporate into the model, and in order to solve this problem all the formulae were included in the model and the choice was left to the user.

The amount of subsidence compensation for structural damages varies from rural to urban areas and there was a problem of considering the average value in the model. Finally, it was accounted for as an input variable in the program.

The pillar extraction cost program will provide an engineer or student a tool to predict the total pillar extraction cost, optimize pillar dimensions to minimize the cost of pillaring, select the degree of surface damage, and provide management information on the relative attractiveness of a pillar extraction project when compared to other alternative methods of total extraction such as longwall and shortwall mining techniques.

Before the size of the coal pillar is computed, the maximum safe roof span of entries or room width is determined. As outlined previously there are many theoretical methods available to determine entry roof span. In addition to ground control considerations, roof span can be selected so as to achieve the highest possible productivity and percentage extraction. In practice, roof spans generally range from 16' to 22' with 18' to 20' being the most common. Actual dimensions are usually based on equipment requirements and experience, rather than computation.

4.1 SAFE PILLAR DIMENSION MODEL

The model is designed to compute safe pillar dimensions for a given geologic and mining environment. The variables used in pillar design are as follows:

--- Depth of coal seam below surface

--- Coal seam thickness (or working height in case of thick coal seam)

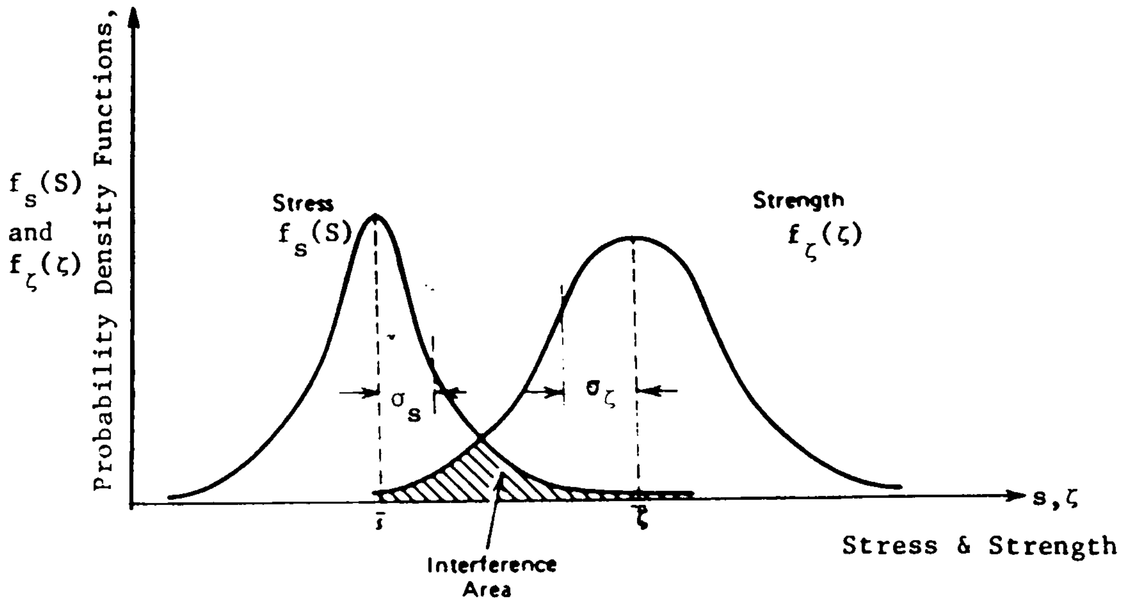
--- Room width

--- Coal strength

--- Safety factor

The conventional deterministic design approach, which gives no indication of failure probability of the coal pillars, is not adequate from a reliability standpoint. Hence, another design methodology that does consider the probabilistic nature of the design has been used. Probabilistic design methodology determines both the pillar stress and pillar strength distributions and expresses the pillar stability reliability as a function of the stress and strength distributions. Failure probability and reliability of the coal pillars depend only on the interference of the stress and the strength distributions; thus, only this information is needed to compute the failure probability and reliability. The interference is illustrated in Figure 4.1 where the shaded portion is the interference-area, indicative of probability of failure.

If the probability density function for the pillar stress (s) be denoted by $f_S(.)$, and that for pillar strength (ζ) by $f_\zeta(.)$, as shown in Figure 4.1, then the probability of pillar strength being greater than pillar stress or the probability that the coal pillars will be



Pillar Stress - Pillar Strength Interference.

Figure 4.1

stable and safe, will be (Kapur, 1977)

$$P(\zeta > s) = P(\zeta - s > 0)$$

$$= \int_{-\infty}^{\infty} f_s(s)\left\{\int_{s}^{\infty} f_\zeta(\zeta)d\zeta\right\}ds$$

or,

$$P(\zeta > s) = \int_{-\infty}^{\infty} f_\zeta(\zeta)\left\{\int_{-\infty}^{\zeta} f_s(s)ds\right\}d\zeta$$

As previously outlined, the variation in coal strength is random variation (Canmet, 1977). In the absence of a large amount of coal strength test data, the average value of coal strength can be used to design coal pillars. A deterministic equation is used to compute coal pillar stress,

$$s = \frac{1.1\ H}{1-e}$$

where,

H = Depth of overburden in feet,

e = extraction ratio.

The derivation of the stress equation is based on the assumptions that the weight of the overburden increases by 1.1 lb/inch^2 for every foot below surface and that the total weight is carried by the pillars in equal proportions.

The equation for the probability of pillar strength being greater than pillar stress reduces to

$$P(\zeta > s) = \int_{s}^{\infty} f_\zeta(\zeta)d\zeta$$

for a constant value of pillar stress, s. Hence only coal pillar strength variation and probability distribution has been accounted for in the model. If enough coal strength test data are available, the concepts of probability and Monte Carlo simulation are used to generate coal strength value.

The probability distribution of a random variable may be defined empirically or through one of many well known probability distributions. Empirical distributions are usually simple to deal with in generating coal strength value using simulation modeling, but may present difficulties from a computational point of view. In many cases we may fail in an attempt to describe the behavior of a random variable through a well-known distribution and thus be forced to use an empirically derived probability distribution. However, where the behavior of the random variable can be adequately characterized by a known probability distribution, it will usually be convenient and useful to do so.

Coal strength test data can be plotted in the form of a histogram and its shape can be compared with theoretical probability distributions to select the one that looks most similar to experimental shape. The parameters of the distribution are estimated using the observed data, and the distribution is tested using Chi-square method. Then the relevant random number generator that has been outlined previously is selected to generate the coal strength.

A computer program has been developed to generate compressive strength of coal. The flowchart of the program is depicted in Figure 4.2 and Appendix A. The user has the option either to choose one of the probability distributions to generate coal strength by supplying a single digit code, or to use the average value of compressive strength. The following probability distributions (Canmet, 1977) are incorporated into the computer program:

Code		Probability Distribution
1	-	Exponential
2	-	Uniform
3	-	Normal
4	-	Empirical

A list of input variables is shown in Table 4.1. If the empirical distribution is selected to generate compressive strength, the values of cumulative probabilities and corresponding strengths are input. The input data set requires the value of the parameters of a particular distribution. The parameters of the distribution are

COMPRESSIVE STRENGTH GENERATOR

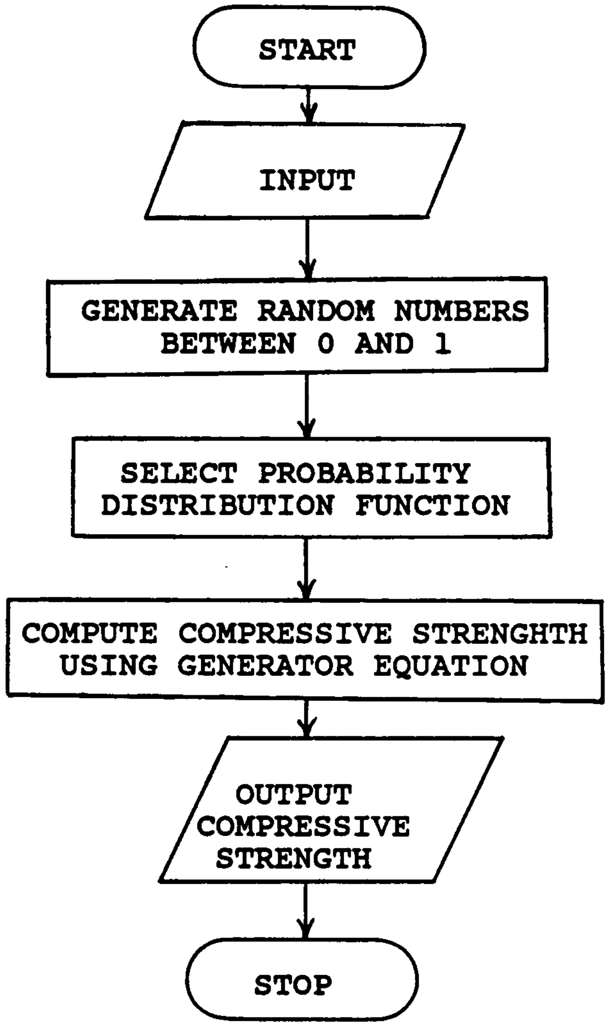


Figure 4. 2

calculated by using the sample data of coal strength.

TABLE 4.1: INPUT DATA

<VARIABLE>	DEFINITION
<CODE>	Code to select a probability distribution.
<SEED>	Any five digit number to generate random numbers.
<LAMDA>	Parameter of exponential distribution: Inverse of average compressive strength.
<MINCOM>	Parameter of uniform distribution: Minimum value of coal strength obtained from sample data.
<MAXCOM>	Parameter of uniform distribution: Maximum value of coal strength obtained from sample data.
<MUCOM>	Parameter of normal distribution: Average value of coal strength obtained from sample data.
<SIGCOM>	Parameter of normal distribution: Standard deviation value of coal strength.
<LCOM(N)>	Compressive strength values for empirical distribution.
<LPROB(N)>	Cumulative probabilities of compressive strengths LCOM(N) for empirical distribution.
<NA>	Number of input data for empirical distribution.

The program reads the input data and goes through the decision statement to select probability distribution. Then it generates the random numbers using the seed value and substitutes the generated random numbers into a generator equation to compute compressive strengths of coal.

The output of the computer program is printed and used to prepare the input data base for the main computer program or pillar extraction cost simulator (PILCOST), which will be discussed later.

The probability of occurrence of compressive strengths equal to or less than a given value, S, can be computed by summing the area under the probability density curve up to S value. If the cumulative distribution function curve of compressive strength is plotted, then it is possible to read the probability of occurrence directly from the curve.

Once the coal strength values are generated, these are used to compute coal pillar strengths by selecting a pillar strength formula.

The general form of coal pillar strength equation is incorporated into the model.

$$\text{Strength} = C + K.W^{\alpha}.h^{\beta}$$

where C, K, α, and β are the coefficients of equation.

This generalization is valid for the following formulae depending upon the selected coefficients of pillar strength:

1) Holland-Gaddy Formula
2) Morrison-Corlett-Rice Formula
3) Obert's Formula
4) Bieniawski's Formula
5) Skelly's Formula
6) Salamon's Formula

<u>Pillar Load:</u>

Pillar load depends on density of overburden rocks. Since the weight of the overburden increases by 1.1 lb/square inch for every foot below the surface, the pillar load equation as used in Safe Pillar Dimension Model is,

$$\text{Load} = 1.1 * \text{Depth} * \left(1 + \frac{\text{Room Width}}{\text{Pillar Width}}\right)^2 .$$

The safety factor is defined as the ratio of strength to load.

$$\text{Safety Factor} = \text{Strength/Load}$$

Pillar Width considered here is edge to edge for square shaped coal pillars.

Safe pillar width (edge to edge) for a given geologic and mining environment can be determined if the values of other variables used in the equations mentioned above are known. Iteration method is used to compute the minimum value of pillar width for a desired safety factor.

4.2 SELECTION OF PILLAR EXTRACTION METHOD

Once the minimum safe pillar dimension is determined, the development percentage extraction can be computed using the equation

$$\% \text{ Extraction} = \left(1 - \left(\frac{\text{Pillar Width}}{\text{Pillar Width + Room Width}}\right)^2\right) * 100.$$

The selection of the particular pillar extraction method is based on the safe pillar dimension. Details of pillar extraction methods have been discussed in the second chapter. Outside Lift (Single cut, and Double cuts), Modified Split & Fender, Christmas Treeing, Split and Fender, and Pocket and Fender methods are the common methods used in this model development. Different pillar widths and corresponding pillar extraction methods have been classified into different groups and presented in Table 4.1a.

TABLE 4.1a: PILLAR EXTRACTION METHODS

Group #	Pillar Width Range	Pillar Extraction Method
1	1' - 13'*	Outside Lift (Single Cut)
2	>13' - 22'	Outside Lift (Double Cuts)
3	>22' - 31'	Modified Split & Fender
4	>31' - <34'	Christmas Treeing Method
5	34' - 55'	Split and Fender
6	>55'	Pocket and Fender

* Not permitted in U.S.A.

The analysis of each class of pillar widths and corresponding pillar extraction methods will be presented in the foregoing lists.

Group 1.

Pillar Width Range = 1' to 13'

Pillar Extraction Method = Outside Lift (Single-Cut) (Figure 4.3)

Pillar Extraction = 97% (Pillar Width <= 8')
= 96% (8'<Pillar Width<=10')
= 95% (Pillar Width = 11')
= 94% (Pillar Width > 11')

Total number of timber props or posts required to extract one pillar = 28 (This includes the breaker posts, and turn or finger posts)

Total number of timber-head-boards required to

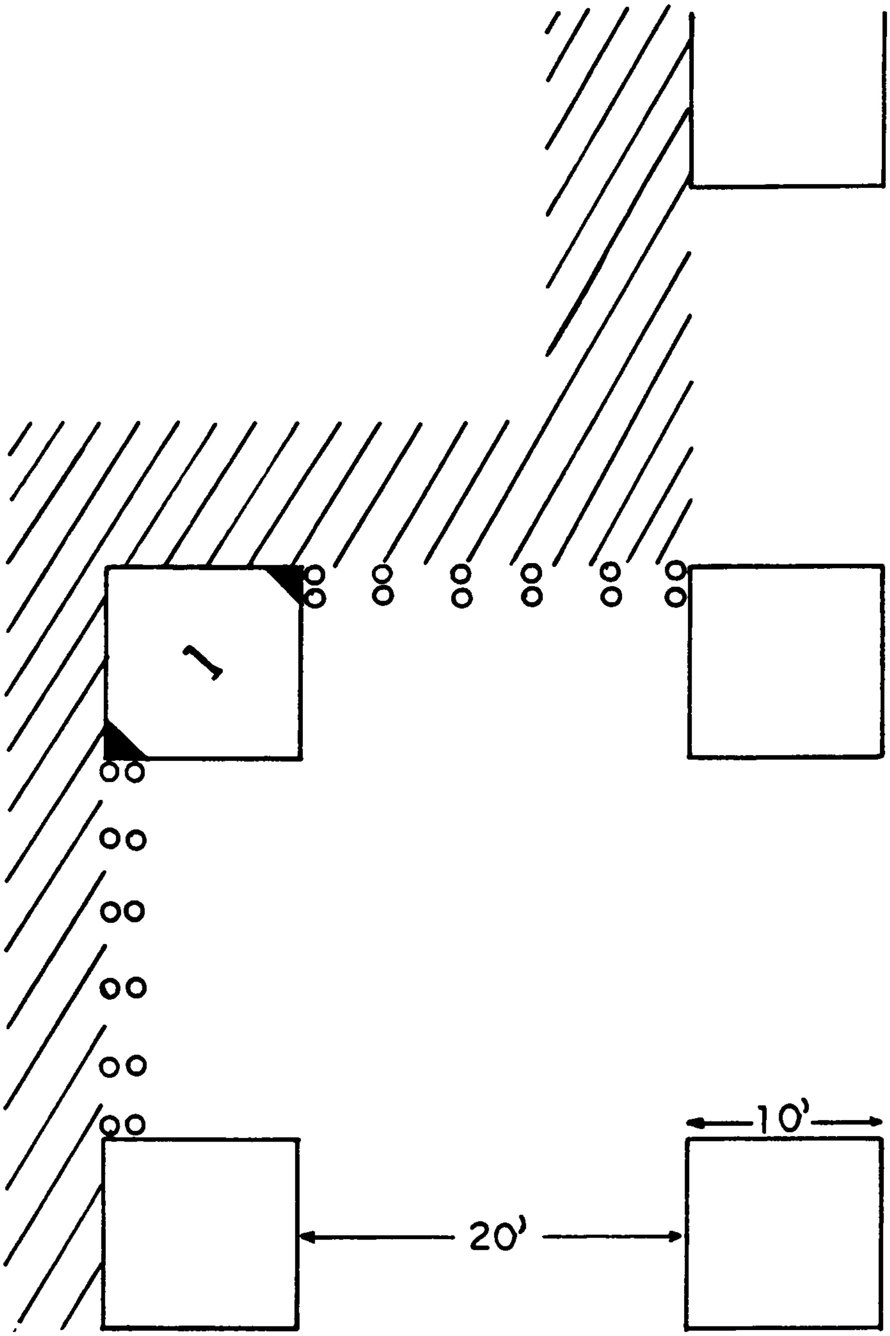


Figure 4. 3 OUTSIDE LIFT (Single Cut) METHOD

extract one coal pillar = 28

Total number of Roofbolts required to extract one coal pillar = 0.

Group 2.

Pillar Width Range = >13' to 22'

Pillar Extraction Method = Outside Lift (Double - Cuts)

Pillar Extraction = 93% (Pillar Width <= 18')
= 92% (Pillar Width > 18')

Total number of timber props or posts required to extract one pillar = 38 (This includes the breaker posts, and turn or finger posts)

Total number of timber-head-boards required to extract one coal pillar = 38

Total number of Roofbolts required to extract one pillar = 0.

Group 3.

Pillar Width Range = >22' to 31'

Pillar Extraction Method = Modified Split & Fender (Figure 4.4)

If Pillar Width <= 24' then,

Pillar Extraction = 91%

Total number of timber props required to

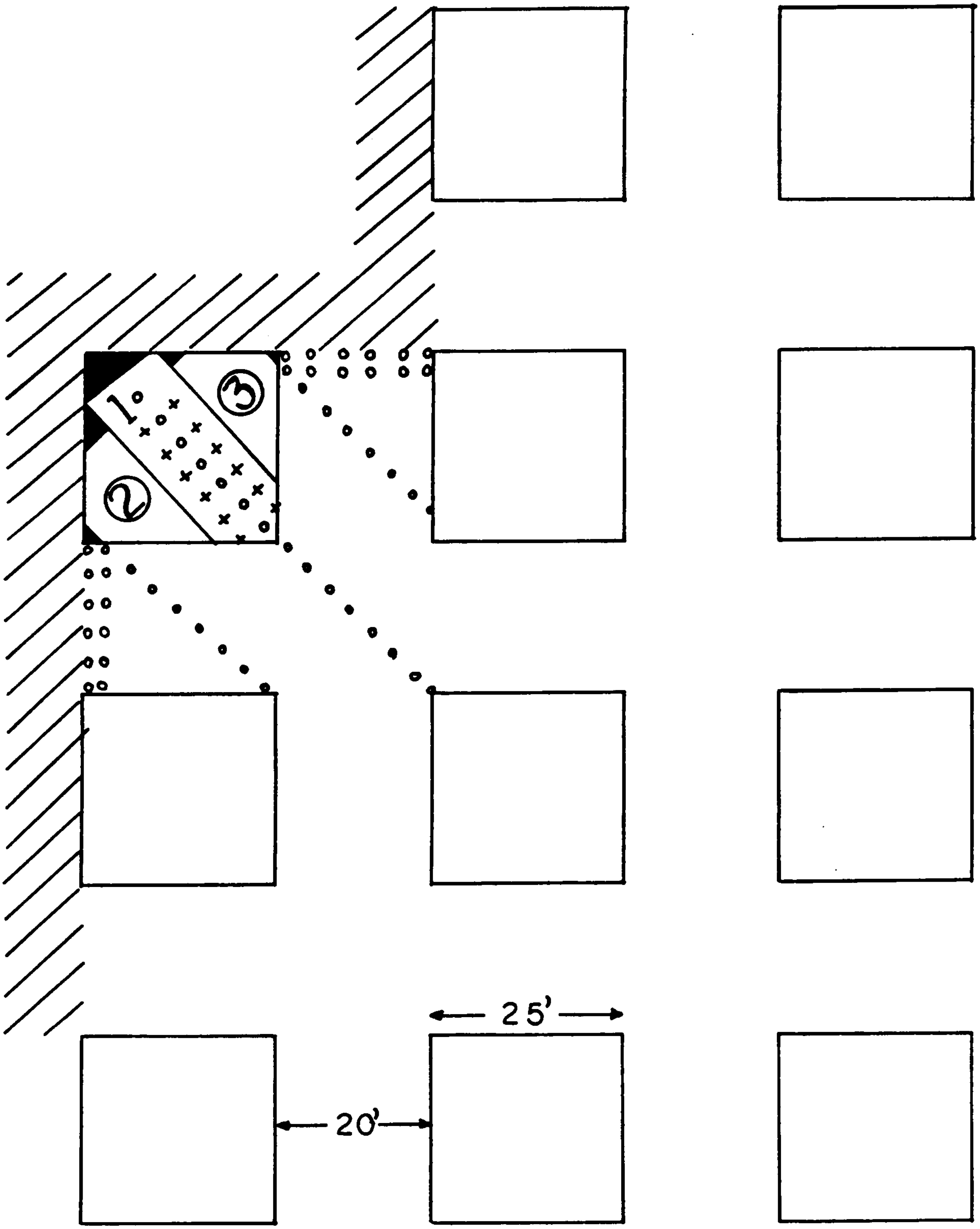


Figure 4. 4 MODIFIED SPLIT-AND-FENDER METHOD

extract one coal pillar = 48

Total number of timber-head-boards required to extract one coal pillar = 48

Total number of Roofbolts required to extract one coal pillar =8

If Pillar Width >24' and <= 27' then,

Pillar Extraction =90%

Total number of timber props required to extract one coal pillar = 49

Total number of timber-head-boards required to extract one coal pillar = 49

Total number of Roofbolts required to extract one coal pillar = 10

If Pillar Width >27' and <=30' then,

Pillar Extraction = 89.5%

Total number of timber props required to extract one coal pillar = 50

Total number of timber-head-boards required to extract one coal pillar = 50

Total number of Roofbolts required to extract one coal pillar = 12

If Pillar Width =31' then,

Pillar Extraction = 89%

Total number of timber props required to

extract one coal pillar = 51

Total number of timber-head-boards required to extract one coal pillar = 51

Total number of Roofbolts required to extract one coal pillar = 14

Group 4.

Pillar Width Range = >31' to <34'

Pillar Extraction Method = Outside-Lift Christmas -Treeing.

Pillar Extraction = 86%

Total number of timber props required to extract one coal pillar = 46 (This includes the breaker props, and turn or finger props)

Total number of timber-head-boards required to extract one coal pillar = 46

Total number of Roofbolts required to extract one coal pillar = 0.

Group 5.

Pillar Width Range = 34' to 55'

Pillar Extraction Method = Split and Fender

Pillar Extraction = 90%

If Pillar Width =34' then,

Total number of timber props required to

extract one coal pillar = 112

Total number of timber-head-boards required to extract one coal pillar = 112

Total number of Roofbolts required to extract one coal pillar = 40

If Pillar Width >34'and <=38' then,

Total number of timber props required to extract one coal pillar = 112

Total number of timber-head-boards required to extract one coal pillar = 112

Total number of Roofbolts required to extract one coal pillar = 45

If Pillar Width >38 and <=43' then,

Total number of timber props required to extract one coal pillar = 112

Total number of timber-head-boards required to extract one coal pillar = 112

Total number of Roofbolts required to extract one coal pillar = 50

If Pillar Width >43' and <=46' then,

Total number of timber props required to extract one coal pillar = 126

Total number of timber-head-boards required to extract one coal pillar = 126

Total number of Roofbolts required to extract one coal pillar = 50

If Pillar Width >46'and <=50' then,

Total number of timber props required to extract one coal pillar = 126

Total number of timber-head-boards required to extract one coal pillar = 126

Total number of Roofbolts required to extract one coal pillar = 55

If Pillar Width >50' and <=54' then,

Total number of timber props required to extract one coal pillar = 126

Total number of timber-head-boards required to extract one coal pillar = 126

Total number of Roofbolts required to extract one coal pillar = 60

If Pillar Width = 55' then,

Total number of timber props required to extract one coal pillar = 126

Total number of timber-head-boards required to extract one coal pillar = 126

Total number of Roofbolts required to extract one coal pillar = 65.

Group 6.

Pillar Width Range = >55'

Pillar Extraction Method = Pocket and Wing

If Pillar Width >55' and <=60' then,

Pillar Extraction = 84%

Total number of timber props required to extract one coal pillar = 186

Total number of timber-head-boards required to extract one coal pillar = 186

Total number of Roofbolts required to extract one coal pillar = 135

If Pillar Width >60' and <=65' then,

Pillar Extraction = 82%

Total number of timber props required to extract one coal pillar = 200

Total number of timber-head-boards required to extract one coal pillar = 200

Total number of Roofbolts required to extract one coal pillar = 155

If Pillar Width >65' and <=73' then,

Pillar Extraction = 81%

Total number of timber props required to extract one coal pillar = 217

Total number of timber-head-boards required to extract one coal pillar = 217

Total number of Roofbolts required to extract one coal pillar = 190

If Pillar Width >73' and <=78' then,

Pillar Extraction = 80%

Total number of timber props required to extract one coal pillar = 233

Total number of timber-head-boards required to extract one coal pillar = 233

Total number of Roofbolts required to extract one coal pillar = 205

If Pillar Width >78' and <=85' then,

Pillar Extraction = 79%

Total number of timber props required to extract one coal pillar = 314

Total number of timber-head-boards required to extract one coal pillar = 314

Total number of Roofbolts required to extract one coal pillar = 230

The methodology of selecting the pillar extraction method is incorporated into the pillar extraction cost model.

4.3 COMPUTATION OF PARAMETERS RELATED TO PILLAR EXTRACTION

Coal reserve is based on 1,800 tons of coal per acre -foot (weight of one-foot cube of coal times number of square feet in one acre is equal to 1800 tons) and a base model case of a five-foot thick coal seam is considered for developing the pillar extraction cost model. The surface area required to provide coal resources necessary to sustain a given period of development operation can be computed using the values of development percentage extraction and seam thickness.

Amount of Coal per acre = 1800*5 tons
(for a 5' thick seam) = 9000 tons

Total amount of coal reserve = 1800*Height(in feet)
per acre (Virgin Property)

Tons mined per acre from Development sections = 1800*
Development Extraction Ratio*Height

Total Acreage Required = Yearly production*Mining
for Development Life/Tons mined per acre
from development sections.

The extra amount of coal mined during pillar extraction or retreat mining is determined using the value of retreat mining percentage extraction.

Extra coal mined during retreat mining = 1800*Total acreage required for development * Height* (Pillar Extraction Ratio - Develop. Extraction Ratio)

Total coal mined by extracting one Coal Pillar =

(Retreat Mining Extraction Ratio - Development Extraction Ratio) * Width^2 * Height * 86 / 2000) / (1 - Development Extraction Ratio).

Total Number of Coal Pillars to be Extracted =

Extra coal mined during Retreat Mining / Coal mined by extracting one coal pillar.

Extended Life of Mine in Years =

Extra coal mined during Retreat Mining / (Ratio of Pillar Extraction to Development Productivities * Yearly Development Production).

Extended Life of Mine in Months =

12 * Extended Life of Mine in Years.

Total Life of Mine in Years =

Development Life in Years + Extended Life in Years.

4.4 COST MODEL

The extended life of the mine is evaluated by the model as discussed in a previous section of this chapter and is used to compute operating costs, and the depreciation cost for pillar extraction based on net-present-value or discount-cash-flow method. As most of the cost data available from USBM reports and other sources are based on costs for mid-1977, the operating costs and other future investments are discounted to the base case of mid-1977.

The cost model developed will compute various operating, depreciation, and subsidence compensation costs for the base case of a 5-foot thick coal seam. The costs will be adjusted and updated for any seam thickness using the sensitivity adjustment factors to the base case of mid-1977. The assessment of retreat mining production costs has been discussed in detail in the third chapter.

The yearly cost data are different for different yearly production rates and mine sizes. The cost of retreat mining production can be obtained from the following: 1). previous project production costs, 2). published production cost data, and 3). scale-up of data for similar mines of other capacities. Previous project production costs may be available in the files of the firm interested in building or constructing the mine. Care, however, must be taken in using such data to consider the time that the data was originally obtained. Cost adjustment factors are available to bring such data up to date.

The use of cost adjustment factors has several limitations, and caution is advised in using them. Cost adjustment factors are based on average values; specific cases may be different from the average.

Costs of similar retreat mining operations are on file or available in the literature but may differ in size from the one under study or analysis. With limited cost data a cost capacity factor can be applied to determine the approximate cost of the new retreat mining operation. The unknown cost C_x of a retreat mining operation of size E_x can be related to a known cost C_k of a retreat mining operation of size E_k as follows:

$$C_x = C_k(E_x/E_k)^n.$$

where n = the cost capacity factor. The cost capacity factor has an average value of 0.65 for most mines and equipment used in mines but can vary over a wide range. The factor can be obtained from published data where a compilation has been made for a number of cost estimates for mines varying in size.

Because the cost data are available for particular production rates or mine sizes, such as 1 million tons, a half- million tons, and 150,000 tons per year, and not for every possible production figure, the unknown cost of a required production rate or mine size can be related to those of the known costs of sizes or production rates.

Several yearly production rates and corresponding cost data have been considered, and the closest cost data can be used to compute unknown cost. Three cost groups will be formed in the cost model in order to compute unknown cost. Each group has a varying size range and if the unknown cost of a given size falls within the size range of a group, the known cost figures of that group will be used for cost adjustment. The grouping of sizes or production rates is shown here:

Production Rates Range	Known Cost Data Used for Adjust.
>750,000 tons/year	1 Million tons/year
250,000-750,000 tons/year	500,000 tons/year
<= 250,000 tons/year	150,000 tons/year

Various cost equations have been developed based on the underground cost data (mid-1977) supplied by Katell (1978, 1975, 1974) in USBM reports, and Haskins (1978) in his research paper.

If unit cost parameters (cost per ton of pillar extraction) for various costs are known, the accuracy of the cost estimation process can be maximized by using the unit cost parameters while computing various costs. The total coal production during retreat mining period is multiplied by a unit cost parameter to compute a particular cost.

Formulation of Direct and Depreciation Costs Equations:

The general equations have been formulated to compute direct and depreciation costs. The values of numerous cost coefficients, a_{hi} (i=1,2,3), and base productions, A_i (i=1,2,3), are presented in Table 4.1b.

Direct Labor and Supervision Cost = Production Labor and Supervision Cost + Maintenance Labor and Supervision Cost

Direct Labor & Supervision Cost = $(a_{1i} + a_{2i})$ * Extended Life of Mine in Years * Yearly Development Production / A_i

Rockbolt & Timber Cost = ($5.42 * No. of Timber Props required to extract one pillar * No. of Coal Pillars to be mined during pillar extraction) + (<u>$57.98*3*30</u>(i.e. monthly roof bolting labor cost)* Extended Life of Mine in Months) + ($2 * No. of Rockbolts required to extract one coal pillar * No. of Coal Pillars to be mined during pillar extraction) + (<u>$10.04*3*30</u>(i.e. monthly bolting equipment depreciation) * Extended Life of Mine in Months).

Mining machine Parts Cost = a_{3i} * Extended Life of Mine in Years * Ratio of Pillar Extraction to Development Productivities * Yearly Development Production / A_i

Lubrication & Hydraulic Oil Cost = a_{4i} * Extended Life of Mine in Years * Ratio of Pillar Extraction to Development Productivities * Yearly Development Production / A_i

Rock Dust Cost = a_{5i} * Extended Life of Mine in Years * Ratio of Pillar Extraction to Development Productivities * Yearly Development Production / A_i

Ventilation Supplies Cost = a_{6i} * Extended Life of Mine in Years * Ratio of Pillar Extraction to Development Productivities * Yearly Development

Production / A_i

Bits Cost = a_{7i} * Extended Life of Mine in Years * Ratio of Pillar Extraction to Development Productivities * Yearly Development Production / A_i

Cables Cost = a_{8i} * Extended Life of Mine in Years * Ratio of Pillar Extraction to Development Productivities * Yearly Development Production / A_i

Miscellaneous Cost = a_{9i} * Extended Life of Mine in Years * Ratio of Pillar Extraction to Development Productivities * Yearly Development Production / A_i

Power Cost = a_{10i} * Extended Life of Mine in Years * Ratio of Pillar Extraction to Development Productivities * Yearly Development Production / A_i

Water Cost = a_{11i} * Extended Life of Mine in Years * Ratio of Pillar Extraction to Development

Productivities * Yearly Development Production / A_i

Royalty Cost = a_{12i} * Extended Life of Mine in Years * Ratio of Pillar Extraction to Development Productivities * Yearly Development Production / A_i

Payroll overhead Cost = 0.40 * Direct Labor and Supervision Cost

Union Welfare Cost = a_{13i} * Extended Life of Mine in Years * Yearly Development Production / A_i

Reclamation Fund Cost = a_{14i} * Extended Life of Mine in Years * Yearly Development Production / A_i

Licenses Cost = a_{15i} * Extended Life of Mine in Years * Yearly Development Production / A_i

Depreciation Cost = a_{16i} * Extended Life of Mine in Years

TABLE 4.1b: COST COEFFICIENTS

Coefficients	Production >750,000 tons	Production >250,000 to 750,000 tons	Production <=250,000 tons
a_{1i}	$3,357,600	$1,971,700	$789,800
a_{2i}	$563,900	$406,200	$155,500
a_{3i}	$770,300	$385,200	$115,500
a_{4i}	$305,900	$153,000	$45,900
a_{5i}	$165,000	$82,500	$24,800
a_{6i}	$237,700	$118,900	$35,700
a_{7i}	$150,100	$75,100	$22,500
a_{8i}	$73,100	$36,600	$11,000
a_{9i}	$182,900	$91,500	$27,400
a_{10i}	$506,500	$281,600	$119,900
a_{11i}	$3,000	$1,400	$400
a_{12i}	$200,000	$100,000	$30,000
a_{13i}	$1,443,400	$759,600	$255,800
a_{14i}	$150,000	$75,000	$22,500
a_{15i}	$100,000	$50,000	$15,000
a_{16i}	$1,256,000	$838,100	$296,260
A_i	1 mil.tons	500,000 tons	150,000 tons

Formulation of Indirect Cost, and Taxes and Insurance Equations:

Indirect Cost = 15% of of labor, supervision,& supplies

or,

Indirect Cost = 0.15 * (Direct Labor & Supervision Cost + Rockbolt & Timber Cost + Mining Machine Parts Cost + Lubrication & Hydraulic Oil Cost + Rock Dust Cost + Ventilation Supplies Cost + Bits Cost + Cables Cost + Miscellaneous Cost).

Total Cost = Direct Labor & Supervision Cost + Rockbolt & Timber Cost + Mining Machine Parts Cost + Lubrication & Hydraulic Oil Cost + Rock Dust Cost + Ventilation Supplies Cost + Bits Cost + Cables Cost + Miscellaneous Cost + Power Cost + Water Cost + Royalty Cost + Payroll overhead Cost + Union Welfare Cost + Reclamation Fund Cost + Licenses Cost + Indirect Cost.

Taxes and Insurance = 2% of Total Cost

or,

Taxes and Insurance = 0.02 * Total Cost

Fixed Cost = Taxes and Insurance + Depreciation Cost

Extra Operating Cost due to Thinning of Coal Seam

(To transform from USBM cost model of 6' thick coal seam to a cost model of given seam thickness; i.e. for transforming to a base model 5' thick seam, an extra $175,000 is added to USBM model of 6' seam).

= $175,000 * Extended Life of Mine in Years
* (6 - Height of Coal Seam).

Formulation of Subsidence Compensation Cost Equation:

1) If surface area is a barren area, then
Subsidence Compensation = 0
2) If surface area is an agricultural land, then
Subsidence Compensation = Reduction in value of land
3) If surface area has buildings, then
Value of Building to be replaced on the target date of mining subsidence = Value of Building =
= Construction cost of building in 1913 *
(Building price index based on age /100)
* (1 - Age of Building/100).

Subsidence compensation is computed by choosing one of the two methods listed below,

a) Vennhofen's Method:

Compensation = (Tilt/2) * (Value of Building / 100)

b) Agreed Basis:

If Tilt <= 20, then

Compensation = (Tilt/2)*Value of Building/100

If Tilt > 20, then

Compensation = (10+(Tilt-20))*Value of Building/100

4) If surface area has communication installations, then there are two methods to protect surface installations,

a) Backfilling Method:

Compensation = Cost per ton of backfilling material * Total backfilling material required.

b) Leaving Coal Pillars Underground:

Compensation = Cost per ton of coal * Total tons of coal left underground to protect surface communication installations.

5) If surface area has mixed features such as agricultural land, buildings, communication installations, etc., then compensation for each of the features on the surface due to mining subsidence is computed separately and added to compute total subsidence compensation cost.

Compensation for agricultural land = Reduction in value of agricultural property,

Value of Building = Construction cost in 1913 * (Building Price Index/100) * (1 - Age of Building /100)

One of these methods is selected to compute value of building,

a) Vennhofen's Method:

Compensation for Building = (Tilt/2)*Value of Building/100

b) Agreed Basis:

If Tilt <= 20, then

Compensation for Building = (Tilt/2)*Value of Building/100

If Tilt > 20, then

Compensation for Building = (10+(Tilt-20))* Value of Building/100

One of these techniques is used to protect surface communication installations,

a) Backfilling Method:

Cost of Backfilling material required = Cost per ton of backfilling material* Total backfilling material required to protect surface installation

b) Leaving Coal Pillars Underground:

Dollar Value of Coal Left Underground = Cost per ton of coal left underground* Total tons of coal left underground to protect surface installations

Subsidence Compensation = Compensation for agricultural land + Compensation for building + Cost of backfilling material required or Dollar value of coal left underground to protect surface communication installations.

Formulation of Inflation Allowance Equation:

Final Total Cost = Direct Cost + Indirect Cost + Fixed Cost + Extra operating cost due to thinning of coal seam + Subsidence Compensation Cost

or, Final Total Cost = Total Cost + Fixed Cost + Extra operating cost due to thinning of coal seam + Subsidence Compensation Cost

$$\text{Inflation Allowance} = \text{Final Total Cost} * ((1+.055)^{(\text{Updating Year}-1977)}-1)$$

$$\text{Overall Cost of Pillar Extraction} = \text{Final Total Cost} + \text{Inflation Allowance}$$

$$\text{Cost of Pillar Extraction per ton of Coal} = \text{Overall Cost/Extra coal mined during depillaring}$$

4.5 DEVELOPMENT OF COMPUTER PROGRAM

A computer program has been developed to compute the total pillar extraction cost and to select optimum pillar sizes to minimize the pillar extraction cost using the cost equations developed in the previous section 4.4.

The program is named PILCOST and is written in FORTRAN-77. It runs on the IBM-PC and PCXT microcomputers. PILCOST requires 30,000 bytes of memory. A brief flowchart of the program is shown in Figure 4.5 and a detailed one in Appendix A. A listing of the program is presented in Appendix B.

The program reads input data and goes through different statements, executes, and prints the computation of various variables. If the user is not interested in built-in cost parameters, the program has the option to receive the unit

FLOWCHART OF PILLAR EXTRACTION COST SIMULATOR (PILCOST)

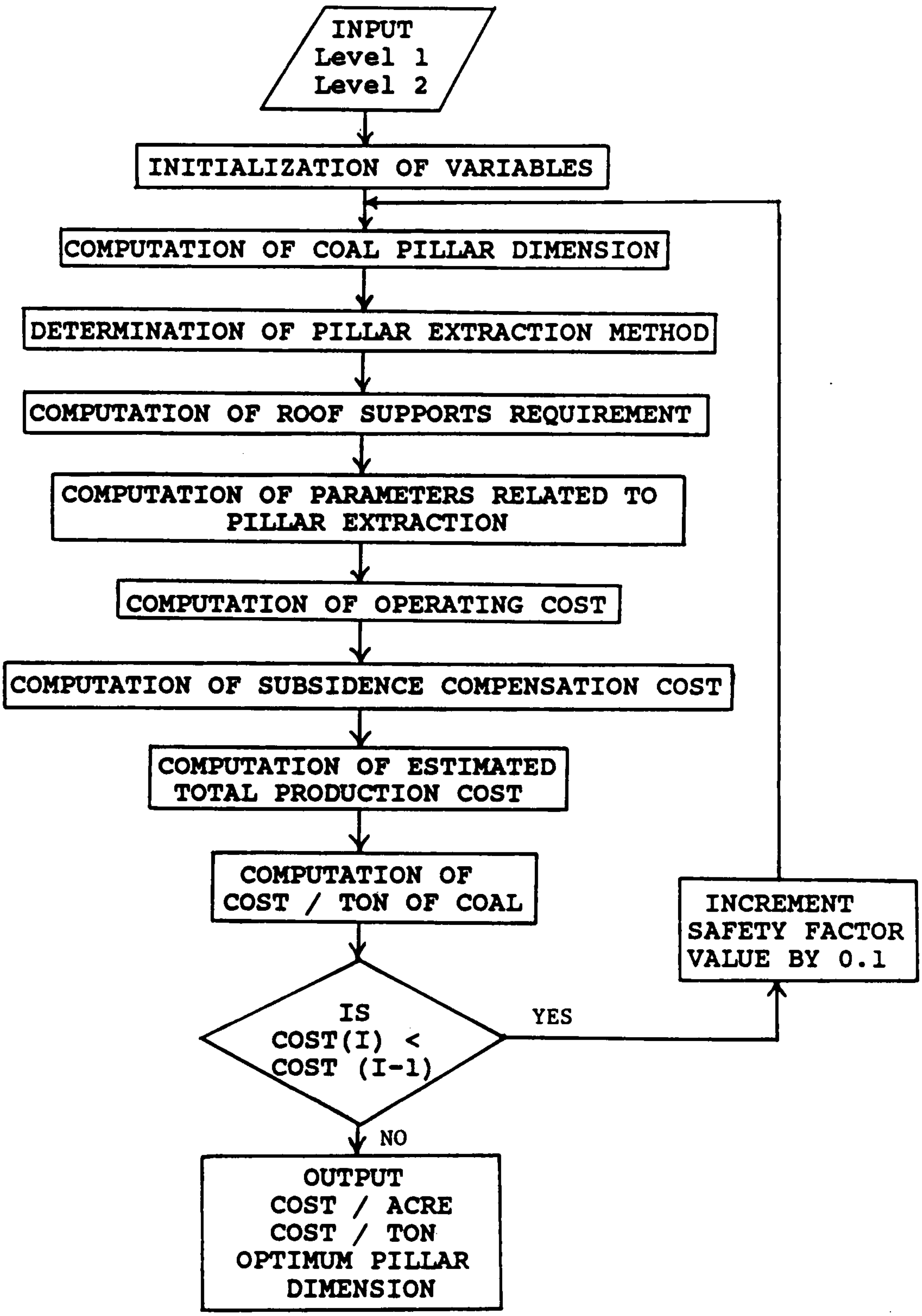


Figure 4. 5

cost parameters supplied by the user and can compute the production cost based on the user's data. The program checks the code for type of cost parameters and if user's input cost parameters are to be utilized to compute production cost the program calls the subroutine USER. Different costs are computed and the control comes back to the main program. Then the program calls the subroutine SUBSID where the subsidence compensation cost is computed and printed. The control comes back to the main program and the rest of the statements of the main program are executed and printed.

There are four main sections of the PILCOST program:

1. Computation of Safe Pillar Dimension
2. Selection of Pillar Extraction Method
3. Computation of Parameters Related to Pillar Extraction
4. Computation of Various Costs.

The first section of the PILCOST program computes the safe pillar dimension for a given geologic and mining environment, and a desired safety factor. After reading the input data, the program initializes the values of pillar extraction cost and an array number. It then goes through a loop, which computes the safety factor of the pillars in the development panel for an initial value of pillar width. Then it compares the computed safety factor to a desired safety factor value. If the computed value of safety factor is less than the desired safety factor, the pillar width value is incremented by one foot and the program repeats the computation of safety factor procedure. In this way, it continues iterating, and finally computes the minimum value of pillar width for which the computed safety factor value is greater than or equal to the desired safety factor.

Once the program accomplishes the determination of safe pillar dimension, it computes the value of development percentage extraction and prints the result.

The second section of the PILCOST program selects the particular pillar extraction method based on safe pillar dimension. Then the program computes the quantity of roof support supplies required to extract coal pillars using the selected pillar extraction method. It calculates the total number of timber posts, wedges, and rockbolts required to extract one coal pillar, and the maximum retreat mining percentage extraction (overall extraction), and prints the results.

The third section of the PILCOST program computes the total amount of coal reserve per acre (virgin property), tons mined per acre from development panels, and the total surface area required to provide coal resources necessary to sustain a given period of development operation. Once this is accomplished by the program, it computes the extra coal mined during pillar extraction, the total coal mined by extracting one pillar, the total number of coal pillars to be mined during pillar extraction operation, the extended life of mine in years, the extended life of mine in months, and the total life of mine in years. Then the program checks the code value for the type of cost data and based on the value of code the control goes to either subroutine USER or next section to use built-in cost parameters.

The last section of the PILCOST program searches for the closest cost data (if built-in cost data available from USBM reports and coal mines are used), which it uses to determine cost adjustment factors. The program computes estimated Direct, Indirect, and Fixed costs, and the extra operating costs from thinning the coal seam. If the cost parameters supplied by the user are to be used the program calls the subroutine USER and computes the various costs. The control then goes to subroutine SUBSID where the subsidence compensation cost is calculated and printed, comes back to the main PILCOST program, and calculates the final total cost.

The program computes inflation allowance, the overall cost of pillar extraction, and the cost of pillar extraction per acre. Also computed is cost of pillar extraction per ton of coal, which is stored as an array.

Then the program compares the current cost of pillar extraction with the preceding cost array. If the current cost value is less than the preceding cost array value, then the program increments the desired safety factor value by 0.1 (thus increasing pillar width) and the control goes back to the beginning of the loop for computing safe pillar dimension.

It computes a new safe pillar dimension and goes through the rest of the sections of the program and computes a new value of pillar extraction cost. Again it compares the new cost value with the preceding cost array value and repeats the process discussed above. In this way it searches for the minimum pillar extraction cost and corresponding pillar dimension which is the optimum pillar dimension to be selected to design coal pillars for extraction. A sample output is shown in Appendix C.

4.5.1 Subroutine USER

This subroutine computes the pillar extraction cost based on the unit cost parameters supplied by the user. Input data are read from a data file and the various operating costs are computed and printed.

4.5.2 Subroutine SUBSID

Subroutine SUBSID reads input data and searches for the type of surface land above the coal pillars. If coal pillars lie under a barren area then it computes the subsidence compensation as zero. Otherwise, it computes different damages and compensations, and prints the result. If coal pillars are to be extracted under a surface land having agricultural farms, buildings, communication installations, or a combination of these, the user is supposed to provide detailed information about the damages and compensations through input data to the program in order to compute the accurate subsidence compensation cost.

4.5.3 Data Input Procedure

Because equations are used to compute the parameters related to pillar extraction and the various costs used in this study, program input requirements have been kept to a minimum. Three data files are required, one for the PILCOST main program and the other two for subroutines USER and SUBSID. Data files can be created or modified by running the DATA, USEDATA, and SUBSIDAT programs. There are two levels of inputs: Level 1 inputs, which are essential, and level 2 inputs, which are not essential if coal pillars lie under a barren area, and are needed only if the surface land above the coal pillars has agricultural farms, buildings, communication installations, or a combination of these. The DATA and USEDATA programs contain level 1 inputs, whereas the SUBSIDAT program contains level 2 inputs. The formats are given so that the user can enter his data easily in the data files. A list of inputs is shown in Table 4.2 (a & b) and Table 4.3.

TABLE 4.2a: INPUT DATA LEVEL 1

<VARIABLE>	DEFINITION
<DEPTH>	Depth of coal seam below surface,
<HEIGHT>	Height of coal pillar or workable height,
<BORD>	Room or entry width in the panel,
<SAFE>	Desired safety factor,
<RATIO>	Ratio of pillar extraction to development productivities,
<YRPROD>	Yearly development production in tons,
<MNLIFE>	Development life time in years,
<UPYEAR>	Year for updating total production cost from base year mid-1977,
<C>	Coefficient in pillar strength equation: $C + K.W^{\alpha}.h^{\beta}$,
<α>	Power index for width in strength equation,
<β>	Power index for height in strength equation,
<TZ>	Code to choose type of unit cost parameters,
<K(I)>	Coefficient in pillar strength equation.

TABLE 4.2b: INPUT DATA LEVEL 1

<VARIABLE>	DEFINITION
<DLCOS1>	Direct labor cost / ton of coal production,
<MPCOS1>	Machine parts cost / ton of coal production,
<LHCOS1>	Lubrication & hudraulic oil cost / ton of coal production,
<RDCOS1>	Rock dust cost / ton of coal production,
<VSCOS1>	Ventilation supplies cost / ton of coal,
<BTCOS1>	Bits cost / ton of coal production,
<CBCOS1>	Cables cost / ton of coal production,
<MSCOS1>	Miscellaneous cost / ton of coal production,
<PWCOS1>	Power cost / ton of coal production,
<WTCOS1>	Water cost / ton of coal production,
<RLCOS1>	Royalty cost / ton of coal production,
<UWCOS1>	Union Welfare cost / ton of coal production,
<RFCOS1>	Reclamation fund cost / ton of coal,
<LICOS1>	Licenses cost / ton of coal production,
<DPCOS1>	Depreciation cost / ton of coal production,
<YRPROD>	Annual development production in tons,
<RATIO>	Ratio of retreat to development mining production rates.

TABLE 4.3: INPUT DATA LEVEL 2

<VARIABLE>	DEFINITION
<TLAREA>	Code for type of surface land. If code = 1, then rest of level 2 inputs are not required.
<RIVAGP>	Reduction in value of agricultural property,
<CONCOS>	Construction cost of building in 1913,
<BPI>	Building price index based on age of building
<AGEBG>	Age of building,
<TILTCD>	Tilt code to choose one of the methods of computing loss in the value of a building due to subsidence, 1 - Vennhofen's method, 2 - Agreed basis,
<TILT>	Tilt value in mm/Metre,
<CICODE>	Code to choose backfilling method or leaving coal pillars underground to protect surface installations,
<CPTCUG>	Cost per ton of coal left underground,
<TTCOAL>	Total tons of coal left underground to protect surface communication installations,
<CPTBFL>	Cost per ton of backfilling material,
<TTBFIL>	Total backfilling material required.

The formats of input variables are mentioned in the User's Manual shown in Appendix D.

4.5.4 Program Output

The output of the program is shown in Appendix C, and includes the following results:

1. Safe pillar dimension (edge to edge),
2. Development percentage extraction,
3. The pillar extraction method used for the computed pillar size,
4. Retreat mining percentage extraction (maximum),
5. Total timber posts required to extract one pillar,
6. Total timber headboards required to extract pillar,
7. Total # of rockbolts required to extract one pillar,
8. Total amount of coal reserve per acre,
9. Tons mined per acre from retreat mining,
10. Total acreage required for development mining,
11. Extra coal mined during pillar extraction,
12. Total coal mined by extracting one pillar,
13. Total number of pillars to be extracted,
14. Extended life of mine in years,
15. Total life of mine in years (including pillaring),
16. Estimated total production cost:

    -- Direct cost:

        Production and maintenance:

            Labour and supervision

Operating supplies:

Rockbolts and timbers

Mining machine parts

Lubrication and hydraulic oil

Rock dust

Ventilation supplies

Bits

Cables

Miscellaneous

Power

Water

Royalty

Payroll overhead

Union welfare

Reclamation fund

Licenses

-- Indirect cost

-- Fixed cost:

Taxes and insurance, and Depreciation

-- Extra operating cost due to thinning of seam

-- Subsidence compensation cost

-- Inflation allowance

-- Overall cost of pillar extraction,

17. Cost of pillar extraction per acre,

18. Cost of pillar extraction per ton of coal.

5. VALIDATION OF THE MODEL

A major objective of this research was to develop a pillar extraction cost model that exhibits the same problems and behaviour characteristics as the system being studied. Validation of the model means to develop an acceptable level of confidence that inferences drawn from the performance of the model are correct and applicable to the real world system (Shannon, 1975; and Stallard, 1981). If output data from the real system is available, it can be compared with the output data from the pillar extraction cost model, and the agreement between the behaviour of the model and that of the real system can be tested.

The first step usually taken in validating any model is a search for errors in the program's logic (Stallard, 1981; and Schmidt, 1970). This can be done by comparing output from the model with hand-calculated values.

Estimating reasonable costs in valuation of a coal property can be quite difficult. The information one has to work with is usually incomplete, so assumptions must be made regarding many things in the pillar extraction cost model. Consequently, the accuracy of the results of any evaluation is dictated by the degree of accuracy of the cost data used in the analysis. USBM Cost Reports and

Crowther (1985) claim an accuracy of within 25 percent of expected actual costs for a mining operation. Because this falls within the 10-30 percent accuracy range acceptable for cost estimates, the USBM Cost Report is a reasonable source of cost data for this evaluation (Gentry, 1980; and Olson, 1984).

If the output data from the pillar extraction cost model differs from the actual output data from the real system and the difference falls within the 10-30 percent accuracy range, then the pillar extraction cost model is acceptable for preliminary cost estimates (Gentry, 1980; & Crowther,1985).

The pillar extraction cost model can be used as either a deterministic or stochastic model. If all the values in the pillar extraction cost model are assigned fixed values and the operation of the system is governed by exact relationships, the model becomes a deterministic model. It can be easily changed and updated to make it stochastic when sufficient field data for at least one of the variables becomes available. The variable becomes random in the model and the model stochastic. The program's logic check is made simple by using the constant values for random variates. All the calculations in the simulator are verified. Logic checks are also performed on the various process generators as well as the random number generators.

CASE STUDIES

In order to validate the pillar extraction cost model, several mining cases have been studied, including coal mines that existed in the past, and mines that are currently active.

Case study 1 will deal with a previously existing coal mine for which pillar extraction cost data are still available. Case study 2 will discuss the validation of the model using cost data available from an existing coal mine.

5.1 Case Study 1:

Room-and-pillar mining cost data were collected from Beth-Elkhorn's No. 22 mine in Letcher County, near Deane, Kentucky, by Green and Palowitch (1976), and were presented in a USBM report. These data have been used to validate the pillar extraction cost model.

The coal mine used a standard room-and-pillar system with pillar extraction in its B unit. The unit consisted of a 10-man production crew working two production shifts per day. The third shift was a maintenance shift. The following production equipment was used near the face:

Continuous miner - 1

Roof bolting machine - 1

Shuttle cars - 2

Battery operated scoop tractor - 1

Section belt ratio feeder - 1

The thickness of the coal seam varied within a 48" to 54" range. The depth of cover varied from 150' to 800'.

Panel dimension:

Entry width = 20' = Crosscut width

Width of pillar = 30' (edge to edge)

Length of pillar = 55' (edge to edge)

Roofbolts were used throughout the section to support the roof. A spacing of $4^1/2$-feet was used between two roofbolts across the entries, and 4-feet spacing was used between the roofbolts along the direction of advance. The average daily production from the development section was 905 tons, which was the same for the pillar extraction section.

At the completion of the eight-entry development, the pillars were extracted on retreat using the outside lift Christmas-treeing method. The same equipment and personnel used for development were used for pillar extraction. The installed track, trolley wire, conveyor, high-voltage cable, and waterlines were recovered during pillar extraction. Actual room-and-pillar recovery or overall retreat mining percentage extraction averaged about 85 percent.

The 10-man production crews mining coal on both the development and pillar extraction phases of the room-and-pillar mining system represented only the personnel working at the face. The same is true of the 4-man maintenance crew.

Production crew cost = \$578.5/shift,
= \$1.28/ton,

Maintenance crew cost = \$228.8/shift,
= 0.25/ton,

Annual depreciation = \$121,770,
(for face equipments) = \$0.64/ton, (based on 1976 prices)

Material cost = \$1.69/ton,

Utility cost = \$0.14/ton

Fixed Indirect cost per year (Insurance, Property taxes, Transportation & cleaning cost, Mine & payroll overhead costs, etc.) = \$4,000,000/section/year,

As it was an experimental mine, the mine had only one production section.

Variable Indirect cost (Royalties, Excise taxes, Union welfare, etc.) = \$2.34/ton of raw coal

Annual coal production = 905*220 tons,
= 199,100 tons,

Total annual cost = (1.28+.25+.64+1.69+.14)*199,100 +
2.34*199,100 + 4,000,000 -
Transportation & cleaning cost.

Fixed Indirect costs of $4 million also include the cost of transportation and cleaning coal (including waste disposal).

Total investment for a 200,000 ton per year coal feed capacity cleaning plant = $\$11,200,000*\left(\frac{200,000}{1,000,000}\right)^{0.96}$

= $2,229,920

Depreciation per year for 10 years life of plant

= $222,992

Cleaning cost and transportation

= $6.60 per ton * 199,100

= $1,353,880

Transportation & cleaning cost

= $222,992 + $1,353,880

= $1,576,872/year

Total cost of pillar extraction

= $1,262,294 + $4,000,000 - $1,576,872

= $3,685,422

Cost of pillar extraction/ton of coal

= $3,685,422/199,100

= $18.51/ton,

The pillars were designed in 1976 and were of rectangular shape (30'X55' edge to edge). As the pillar extraction cost simulator computes the safe pillar dimension of square-shaped pillars by selecting one of the pillar design equations, an equivalent square-shaped area of the pillar was computed for a given mine section (30*55 = 1650 square feet). The size of the equivalent square-shaped pillar was computed using the cross-sectional area of the pillar (- 40' edge to edge). Several values of coefficients of pillar design equation were tried as input data and the computer program was run each time. By trial and error, the safe pillar dimension was computed to be 40 feet (edge to edge). The cost output data relevant to this pillar size were analyzed.

The other input data for the computer program are shown below:

<u>Level 1 Input:</u>

Yearly Development Mining Production = 199,100 tons,

Development Mining Life = 10 years,

Year for updating the production cost = mid-1976,

Height of coal seam = 4.5',

Room width = 20',

Ratio of pillar extraction to development productivities = 1,

<u>Level 2 Input:</u>

Code for type of surface land = 1 (barren area), Reduction in value of agricultural property = 0,

Construction cost of damaged building in 1913 = 0,

Building price index based on age of building = 0,

Age of building = 0,

Tilt code = 0,

Tilt value in mm/Metre = 0,

Code to choose backfilling method = 0,

Cost per ton of coal left underground = 0,

Total tons of coal left underground to protect surface communication installations = 0,

Cost per ton of backfilling material required = 0,

Total backfilling material required to protect communication installations = 0,

Output:

The computer output is shown below:

Safe pillar dimension is 40 feet,

Development percentage extraction = 55.56,

The pillar extraction method used for this pillar size will be "Split and Fender",

Retreat mining percentage extraction (maximum) = 90.0,

Total timber posts required to extract one pillar = 112,

Total timber headboards reqd. to extract one pillar = 112,

Total # of rockbolts required to extract one pillar

= 50,

Total amount of coal reserve per acre = 8100 tons,

Tons mined per acre from development = 4500.0,

Total acreage required for development = 442.44,

Extra coal mined during retreat mining = 1234420 tons

Total coal mined by extracting one pillar = 239.9 tons,

Total number of pillars to be extracted = 5145,

Extended life of mine in years = 6.2,

Extended life of mine in months = 74.4,

Total life of mine in years (including pillar extraction) = 16.20,

ESTIMATED TOTAL PRODUCTION COST

DIRECT COST:

Production & Maintenance:	
Labor & Supervision	$ 5860859
Operating Supplies:	
Rockbolts and Timbers	$ 4093182
Mining Machine Parts	$ 716099
Lubrication & Hydraulic Oil	$ 284579
Rock Dust	$ 153759
Ventilation	$ 221339
Bits	$ 139499
Cables	$ 68199
Miscellaneous	$ 169879
Power	$ 743379

Water	$	2479
Royalty	$	185999
Payroll Overhead (40% of Payroll)	$	2344343
Union Welfare	$	1585959
Reclamation Fund	$	139499
Licenses	$	92999
INDIRECT COST:		
15% of Labor, Supervision, & Supplies	$	1756109
TOTAL	$	18558160
FIXED COST:		
Taxes and Insurance (2% of Mine Cost)	$	371163
Depreciation	$	1836811
GRAND TOTAL (Fixed, Direct, & Indirect)	$	20766134
EXTRA OPERATING COST DUE TO THINNING OF SEAM (Deviation from base model of 6' thick seam)	$	1627499
SUBSIDENCE COMPENSATION COST	$	0
FINAL TOTAL	$	22393633
INFLATION ALLOWANCE	$	-1187301
OVERALL COST OF PILLAR EXTRACTION	$	21206332

COST OF PILLAR EXTRACTION PER ACRE = $ 48,791,

COST OF PILLAR EXTRACTION PER TON OF COAL = $ 17.00, The cost of pillar extraction per ton of coal as obtained from the computer model was \$17.00. In actual pillar extraction operation, it was \$18.51 per ton. There are several factors that might have caused this difference. The most important are the geologic factors or variation in geologic condition of the actual mine from the the computer model. As the various cost components data vary from area to area up to a certain limit, the overall cost of pillar extraction also varies from mine to mine or from section to section in the same mine.

Difference in result = \$18.51 - \$17.00

= \$1.51/ton

Therefore, Variation in result = 8.16%

This is within the 20-30% limit claimed by Gentry (1980) and 15-25% limit claimed by Crowther (1985).

5.2 <u>Case Study 2</u>:

Pillar extraction cost data were collected from a currently active mine in southwestern Virginia. The coal mine used standard room-and-pillar system with pillar extraction over the entire mine property. Depth of the coal seam below the surface was 450'. The thickness of the coal seam or working height was 67". Entry width was 19' and width and length of pillar were 62'X62' (centre to centre).

Development, pillaring, and subsidence compensation related data are listed as follows:

Development

Annual production from development sections -- 387,684 tons,

Mining life -- 16 years.

Pillaring

Ratio of pillar extraction to development productivities -- 5,

Year of pillar extraction -- 1985,

Pillar extraction method -- Split block and wing, leaving four stumps,

Percentage extraction in pillaring sections -- 60% of a pillar,

Overall percentage extraction of the coal property after pillaring -- 80.76%,

Total coal mined during pillaring -- 3,449,268 tons,

Cost of pillar extraction /ton of coal -- $10.25 (without subsidence compensation).

Subsidence Compensation Related Data

Type of surface land -- Mountainous with some residential.

If surface area has buildings then age and cost of building during construction period -- 12 houses, average age of 20 years,

Average cost of construction -- $22,000.

Tilt value in mm/Metre -- Unknown,

Method used to protect surface installation --

Leaving large coal pillars underground,

Cost/ton of coal left underground to protect surface installation -- $2.50,

Total tons of coal left underground to protect surface installation -- 3,876,840 tons,

Total subsidence compensation or damage protection cost -- $9,692,100

Subsidence compensation or damage protection cost per ton of pillar extraction -- $2.80.

Cost of pillar extraction /ton of coal = $10.25 + $2.80

= $13.05.

(This includes subsidence compensation cost also)

The pillars were designed in the past and were of square shape with an edge length of 43'. The safety factor corresponding to the 43-feet pillar size (edge to edge) was computed and the result was 2.3 for Salamon's equation. The computer program was used to compute pillar extraction cost data and the input data for computer program are shown below:

<u>Level 1 Input</u>:

Depth of coal seam below surface = 450', Seam thickness (or working height) = 5.6',

Room or entry width = 19',

Desired safety factor = 2.3,

C (coeff. in pillar strength formula: $C + K.W^{\alpha}.h^{\beta}$) = 0,

α (power index of width in the formula) = 0.46,

β (power index of height in the formula) = -0.66,

K (coefficient in pillar strength formula) = 1320,

Annual production from development sections = 387,684 tons,

Mining life (development only) = 16 years,

Ratio of pillar extraction to development productivities = 5,

Year of pillar extraction = 1985.

<u>Level 2 Input</u>:

Code for type of surface land = 5,

Reduction in value of agricultural property = 0,

Construction cost of damaged building in 1913 = 0,

Building price index based on age of building = 0,

Age of building = 20 years,

Tilt code = 0,

Tilt value in mm/Metre = 0,

Code to choose backfilling method or leaving coal pillars = 2,

Cost /ton of coal left underground (1977 \$) = \$1.629,

Total tons of coal left underground to protect surface installation = 3,876,840 tons,

Cost /ton of backfilling material required = 0,

Total backfilling material required to protect surface installation = 0.

Output:

The computer output is shown below:

Safe pillar dimension is 43 feet,

Development percentage extraction = 51.9,

The pillar extraction method used for this pillar size will be "Split-and-Fender",

Retreat mining percentage extraction (max.) = 90.0,

Total timber posts required to extract one coal pillar = 112,

Total timber headboards required to extract one coal pillar = 112,

Total number of rockbolts required to extract one coal pillar = 50,

Total amount of coal reserve per acre = 10079 tons,

Tons mined per acre from development = 5231.43 tons,

Total acreage required for development = 1185.71,

Extra coal mined during retreat mining = 4553800 tons

Total coal mined by extracting one pillar = 352.7 tons,

Total number of pillars to be extracted = 12912,

Extended life of mine in years = 2.349 years,

Extended life of mine in months = 28.19 months, Total life of mine in years (including pillar extraction) = 18.349 years.

ESTIMATED TOTAL PRODUCTION COST

DIRECT COST:	
Production & Maintenance:	
Labor & Supervision	$ 4331392
Operating Supplies:	
Rockbolts and Timbers	$ 9301879
Mining Machine Parts	$ 3508247
Lubrication & Hydraulic Oil	$ 1393462
Rock Dust	$ 751376
Ventilation	$ 1082893
Bits	$ 683980
Cables	$ 333338
Miscellaneous	$ 833345
Power	$ 2564700
Water	$ 12750
Royalty	$ 910759
Payroll Overhead (40% of Payroll)	$ 1732556
Union Welfare	$ 1383626
Reclamation Fund	$ 136613
Licenses	$ 91075
INDIRECT COST:	
15% of Labor, Supervision, & Supplies	$ 3332986
TOTAl	$32384977

FIXED COST:

Taxes and Insurance (2% of Mine Cost)	$ 647699
Depreciation	$ 1968892
GRAND TOTAL (Fixed, Direct, & Indirect)	$35001568
EXTRA OPERATING COST DUE TO THINNING OF SEAM	
(Deviation from base model of 6' thick seam)	$ 164446
SUBSIDENCE COMPENSATION COST	$ 6315372
FINAL TOTAL	$41481386
INFLATION ALLOWANCE	$22179537
OVERALL COST OF PILLAR EXTRACTION	$63660923

COST OF PILLAR EXTRACTION PER ACRE = $53,690,

COST OF PILLAR EXTRACTION PER TON OF COAL = $13.97,

This estimated cost of pillar extraction per ton of coal is relevant to an annual inflation rate of 5.5%. In actual pillar extraction operation, it was $13.05 per ton of pillar extraction including subsidence compensation cost.

Difference in result = $13.97 - $13.05

= $0.92/ton

Therefore, Variation in result = 7.05%

This is within the 20-30% limit claimed by Gentry (1980) and the 15-25% limit claimed by Crowther (1985) for the validity of any mining cost model.

Pillar extraction costs have been estimated for various annual inflation rates and presented as follows:

Inflation Rate	Estimated Cost/ton	% Variation
2.0%	$10.67	18.21%
2.5%	$11.09	14.95%
3.0%	$11.54	11.57%
3.5%	$12.00	8.08%
4.0%	$12.46	4.47%
4.5%	$12.95	0.73%
5.0%	$13.46	3.12%
5.5%	$13.97	7.05%

The variations resulting from various annual inflation rates are within the acceptable limits.

6. SENSITIVITY ANALYSIS

Sensitivity analysis is the process by which input data elements in the pillar extraction cost model are varied to determine their impact on the overall cost of pillar extraction, and the cost of pillar extraction per ton of coal. Sensitivity analysis identifies the key criteria affecting the cost model, such as depth, thickness of coal seam, safety factor, ratio of retreat to development mining production rates, yearly development production, and development mining life. The effects on the cost of extracting coal pillars in a room-and-pillar mining system created by changing these variables are displayed in graphical form in Appendix E. Several graphs displayed cover most of the possible geologic and mining conditions encountered in the coalfields.

In the case of development operation, the cost of coal extraction per ton of coal increases with the increasing depth of mining. Several factors, such as the cost of shaft sinking, the cost of ground control at greater depths, and the cost of controlling coal bumps and bursts, increase the cost. The cost of lost coal underground (based on mine property cost) also increases with increasing depth. This cost is accounted for while computing the cost of development. Figure 6.1 depicts the cost of development versus the depth, and Figure 6.2 depicts the dollar value of lost coal underground versus depth.

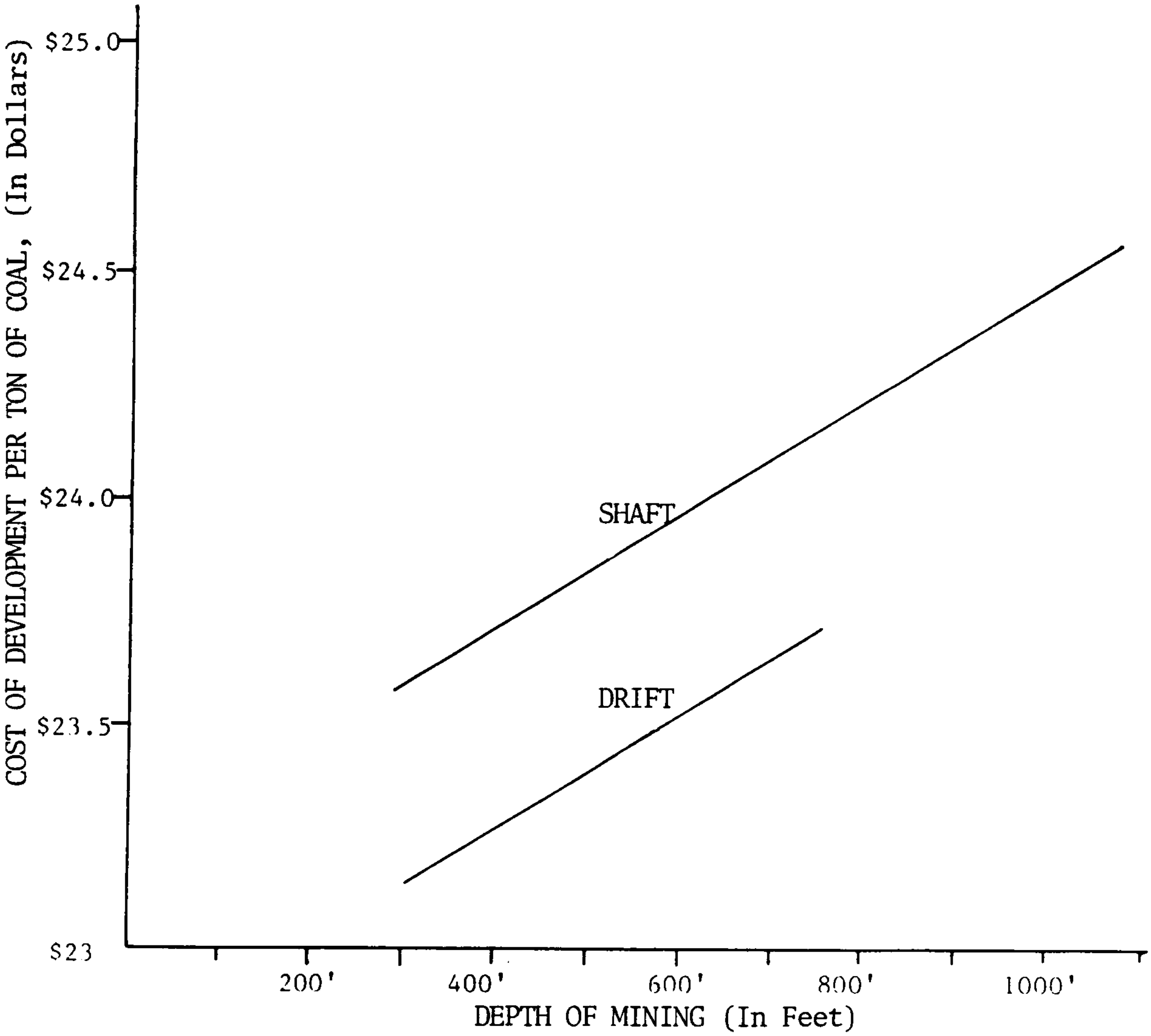
Productivity on a retreat mining section can be significantly higher than on an identical advance section. The cost of pillar extraction per ton of coal decreases as depth increases up to a certain value, remains nearly the same as depth increases further, then increases with increasing depth. Since the number of breaker and turn posts and roofbolts required in the splits and entries to extract one coal pillar is almost the same for shallow to medium depths, the cost of pillar extraction is higher for shallow depths. Figures 6.3 to 6.5 depict the cost of pillar extraction per ton of coal versus depth for various mine sizes, such as 1 million tons per year, 500,000 tons per year, and 150,000 tons per year.

The cost of pillar extraction per acre of mine property increases with increasing depth of mining. This characteristic is depicted in Figure 6.6.

The cost of pillar extraction per ton of coal decreases with the increasing ratio of retreat to development mining productivities. This is shown in Figure 6.7 for ratio values such as 1.00, 1.50, and 2.00.

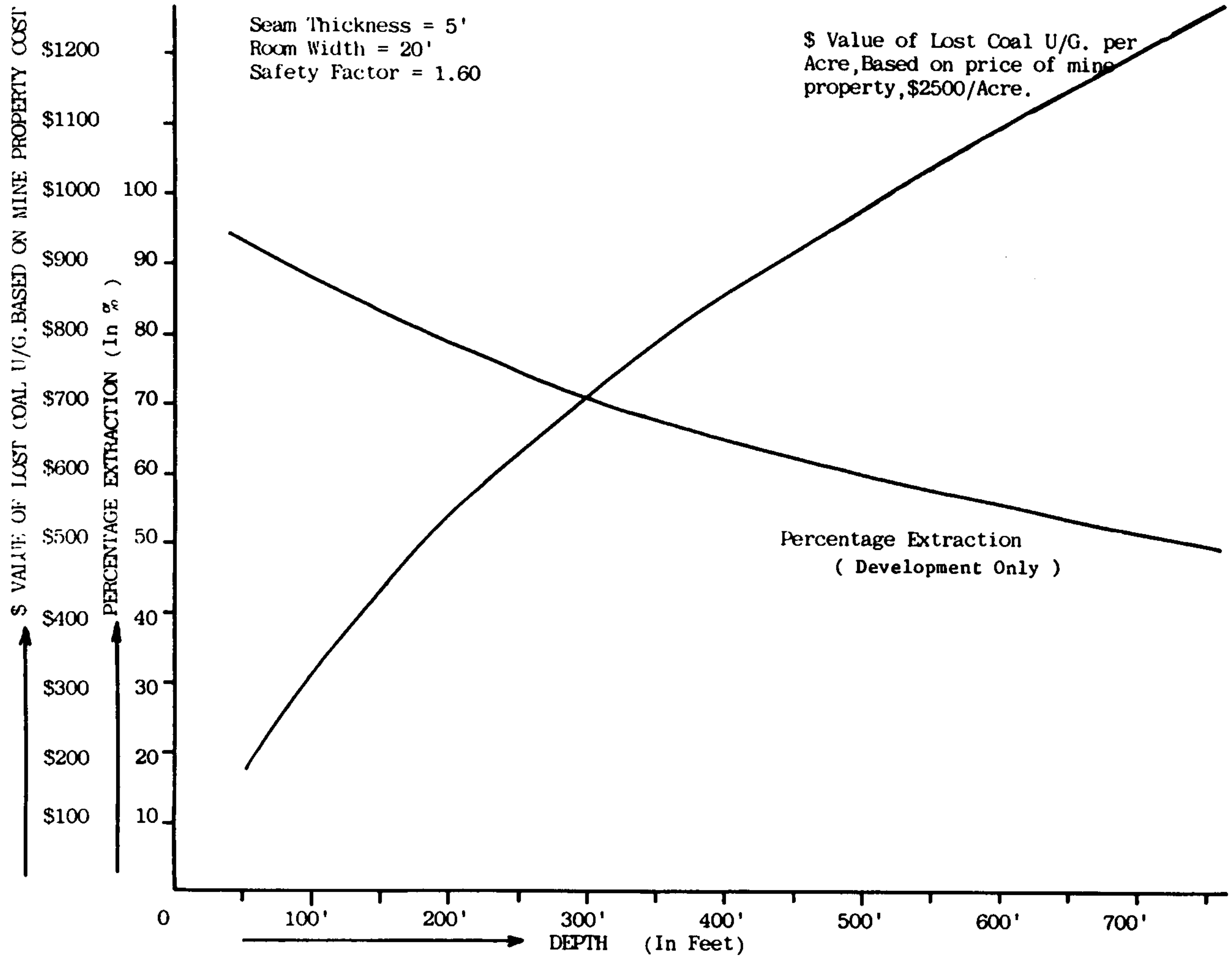
As the desired safety factor used for basic pillar size design increases, the cost of pillar extraction per ton of

ENTRY WIDTH = 20'
SEAM THICKNESS = 6'
YEAR OF PRODUCTION = 1986



Cost of Development versus Depth of Mining (For Shaft & Drift Mines)

Figure 6.1



Dollar Value of Lost Coal Underground versus Depth of Mining.

Figure 6.2

ENTRY WIDTH = 20'

SEAM THICKNESS = 5'

SAFETY FACTOR = 1.6 (Salamon's Equation)

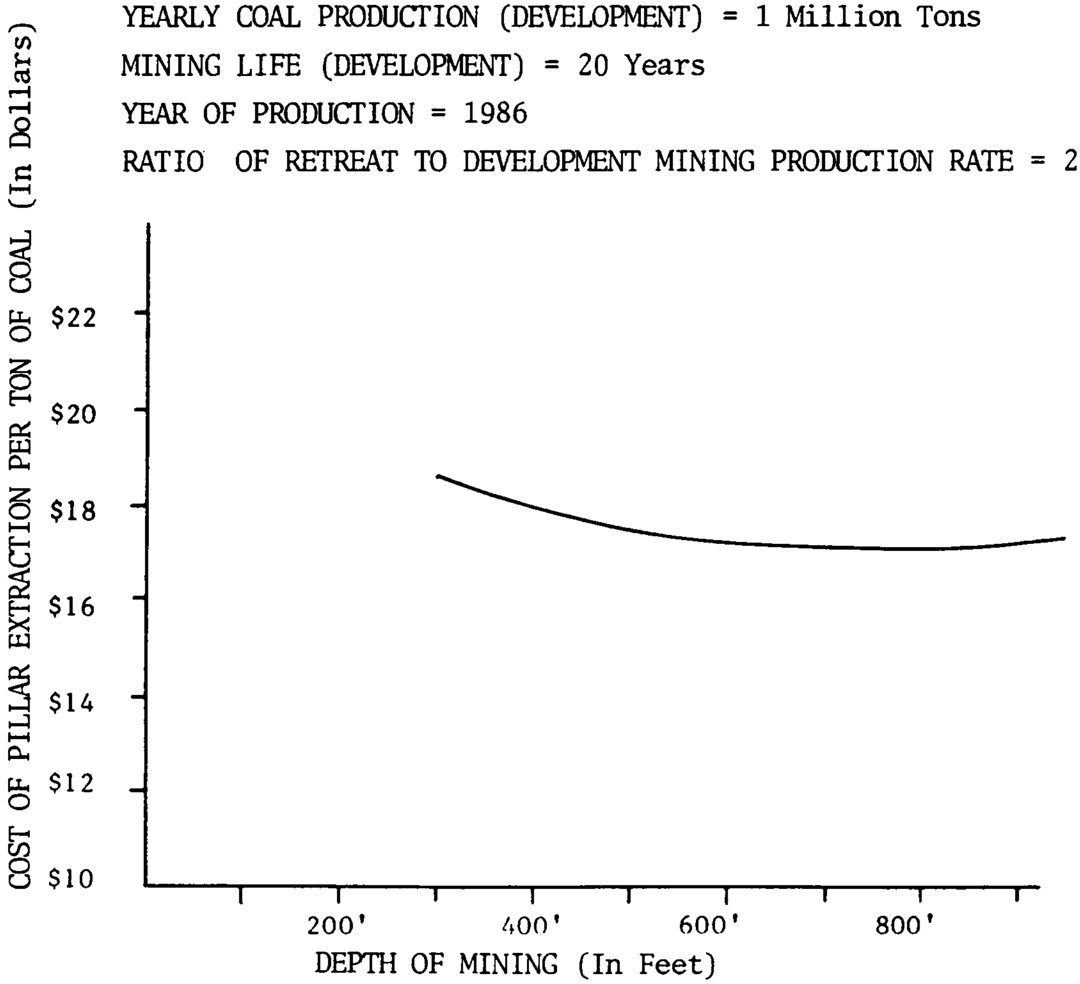
SUBSIDENCE COMPENSATION COST = 0 (Assumed)

YEARLY COAL PRODUCTION (DEVELOPMENT) = 1 Million Tons

MINING LIFE (DEVELOPMENT) = 20 Years

YEAR OF PRODUCTION = 1986

RATIO OF RETREAT TO DEVELOPMENT MINING PRODUCTION RATE = 2



Relationship between Cost of Pillar Extraction Per Ton of Coal and Depth of Mining (Mine Size - 1 Million Ton)

Figure 6.3

ENTRY WIDTH = 20'

SEAM THICKNESS = 5'

SAFETY FACTOR = 1.6 (Salamon's Equation)

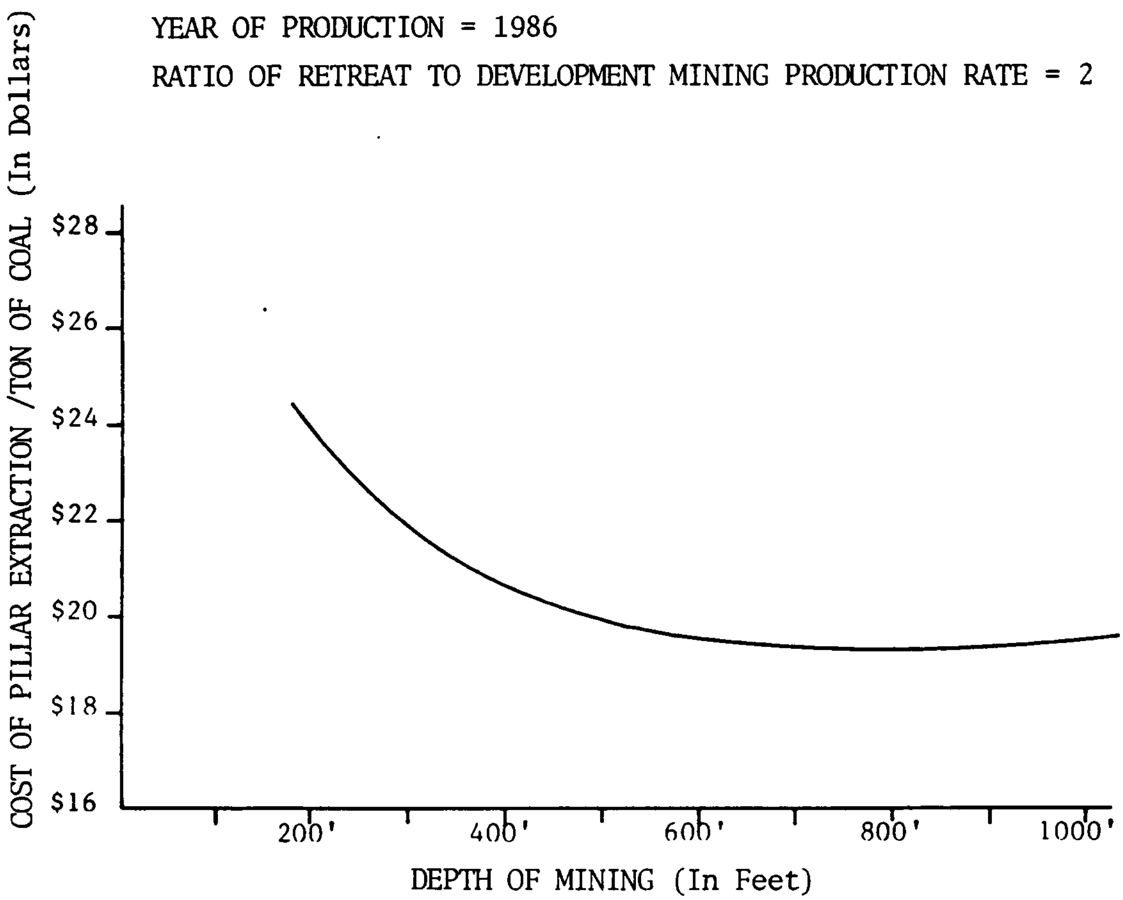
SUBSIDENCE COMPENSATION COST = 0 (Assumed)

YEARLY COAL PRODUCTION (DEVELOPMENT) = 500,000 Tons

MINING LIFE (DEVELOPMENT) = 20 Years

YEAR OF PRODUCTION = 1986

RATIO OF RETREAT TO DEVELOPMENT MINING PRODUCTION RATE = 2



Relationship between Cost of Pillar Extraction Per Ton of Coal and Depth of Mining (Mine Size - 500,000 Tons).

Figure 6.4

ENTRY WIDTH = 20'

SEAM THICKNESS = 5'

SAFETY FACTOR = 1.6 (Salamon's Equation)

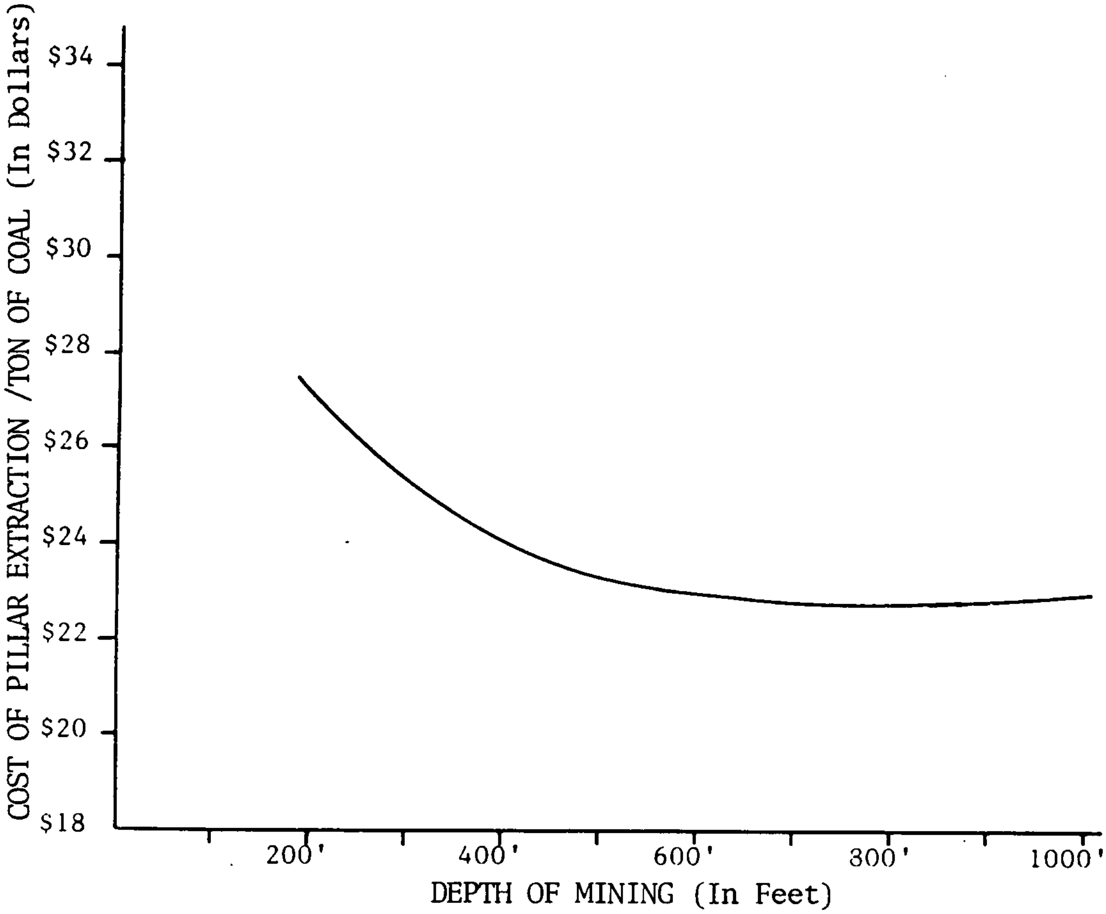
SUBSIDENCE COMPENSATION COST = 0 (Assumed)

YEARLY COAL PRODUCTION (DEVELOPMENT) = 150,000 Tons

MINING LIFE (DEVELOPMENT) = 20 Years

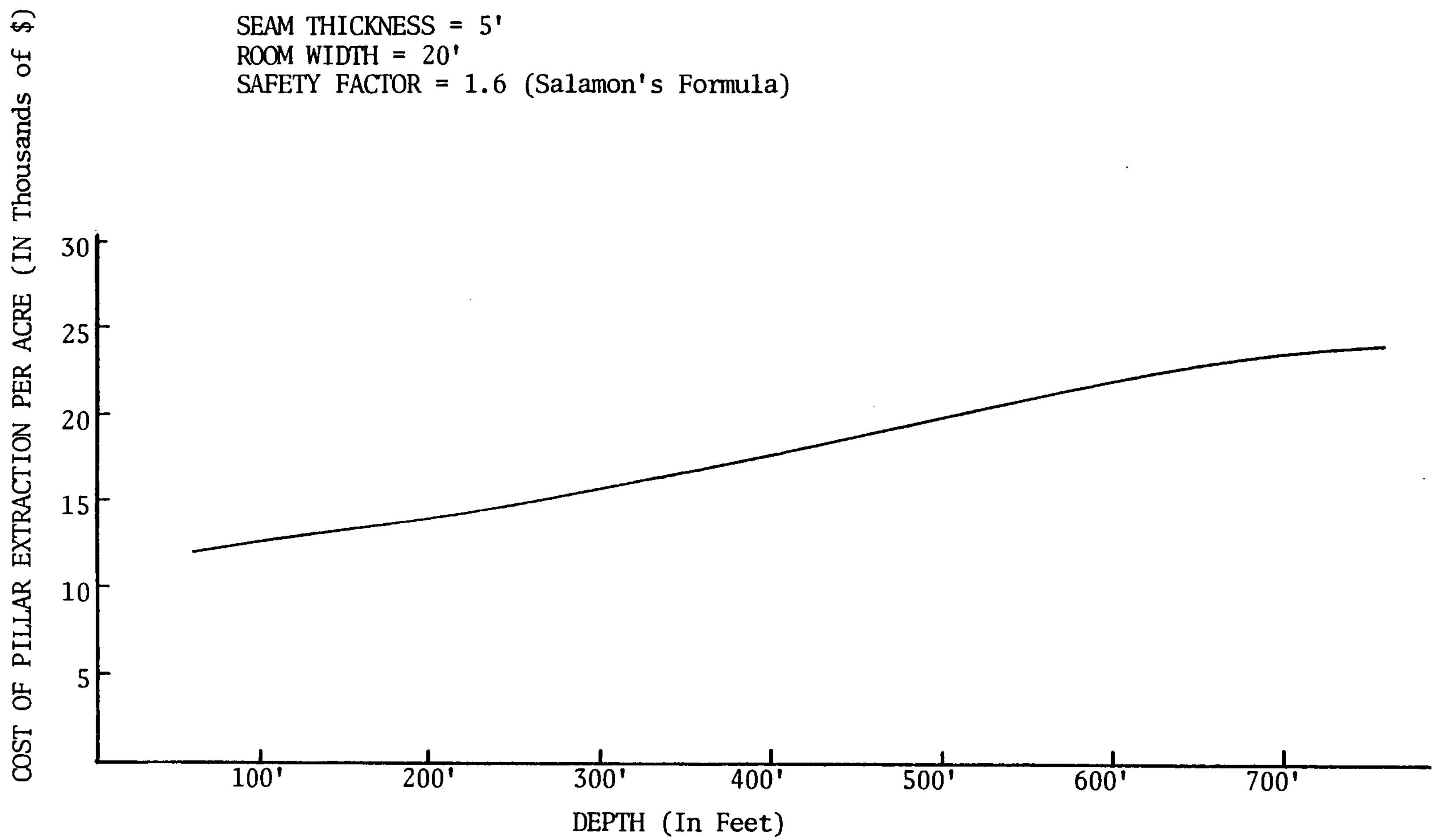
YEAR OF PRODUCTION = 1986

RATIO OF RETREAT TO DEVELOPMENT MINING PRODUCTION RATE = 2



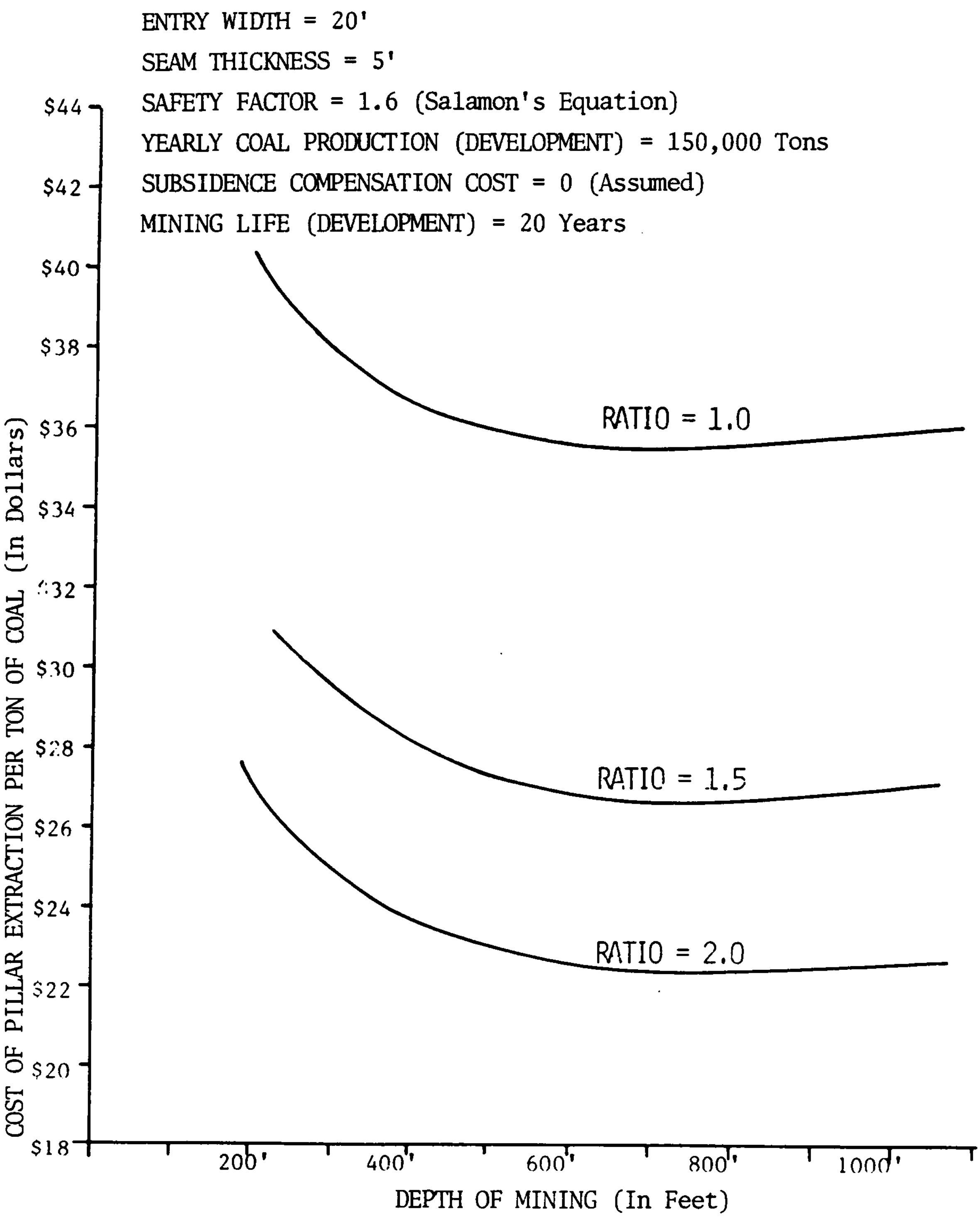
Relationship between Cost of Pillar Extraction Per Ton of Coal and Depth of Mining (Mine Size - 150,000 Tons).

Figure 6.5



Cost of Pillar Extraction Per Acre of Mine Property versus Depth of Mining.

Figure 6.6



Cost of Pillar Extraction Per Ton of Coal versus Ratio of Retreat to Development Mining Production Rates for Varying Depths .

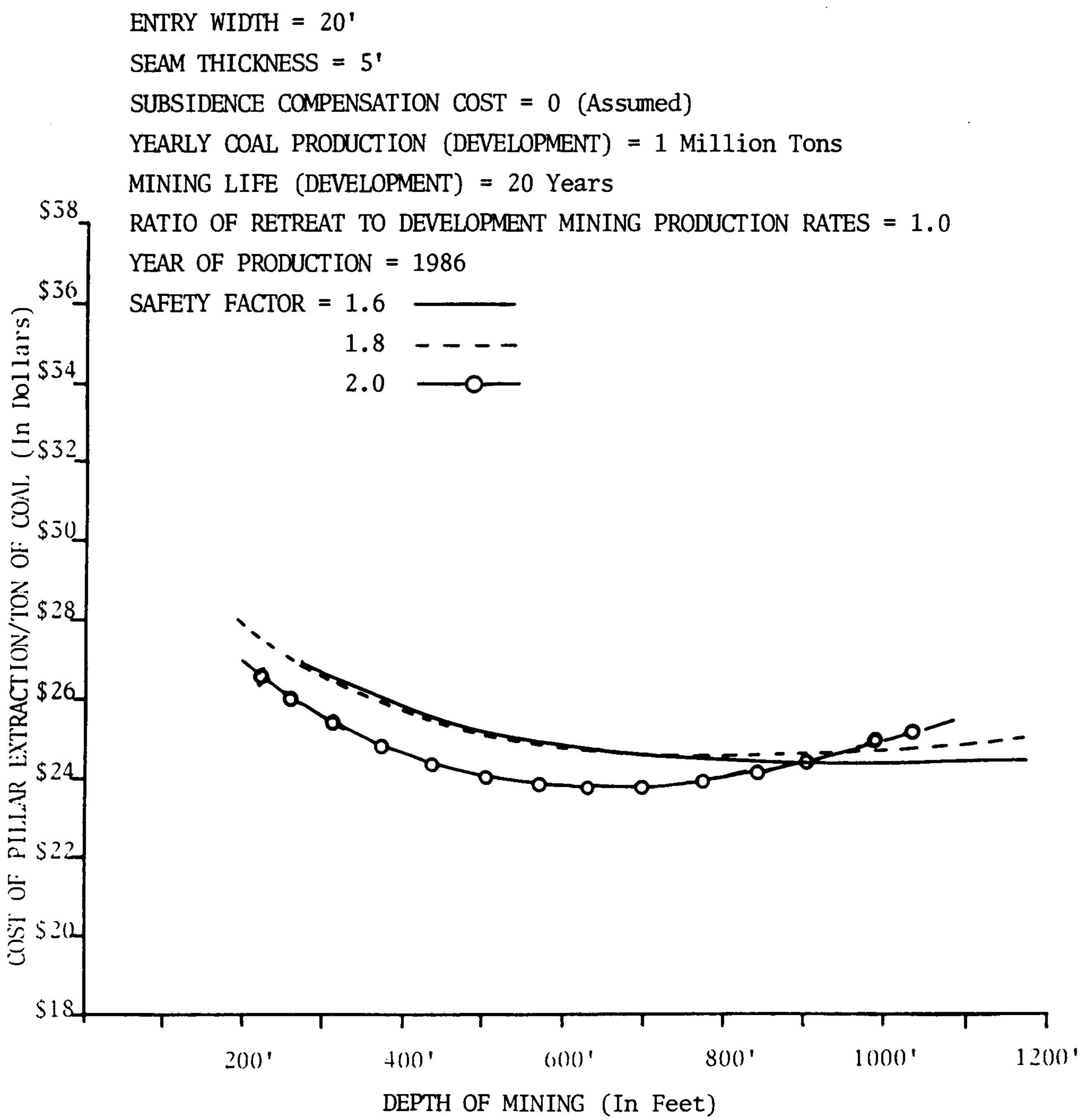
Figure 6.7

coal decreases by a small amount (\$0.5 to \$1.0) for shallow depths, remains the same for further increases in depth up to a certain limit, then increases with increasing depth. Figure 6.8 depicts this relationship.

The cost of pillar extraction per ton of coal decreases with the increasing size of the development mining operation or the capacity of the mine. Figure 6.9 depicts this relationship for mine sizes such as 1 million tons per year, 500,000 tons per year, and 150,000 tons per year. This result indicates that pillar extraction can become economically feasible by increasing the size of the development mining operation. In some cases, the pillar extraction cost may be a little higher than the market price of mined coal and will discourage investment. But if the size of the development mining operation is increased it may be possible to extract the coal pillars economically.

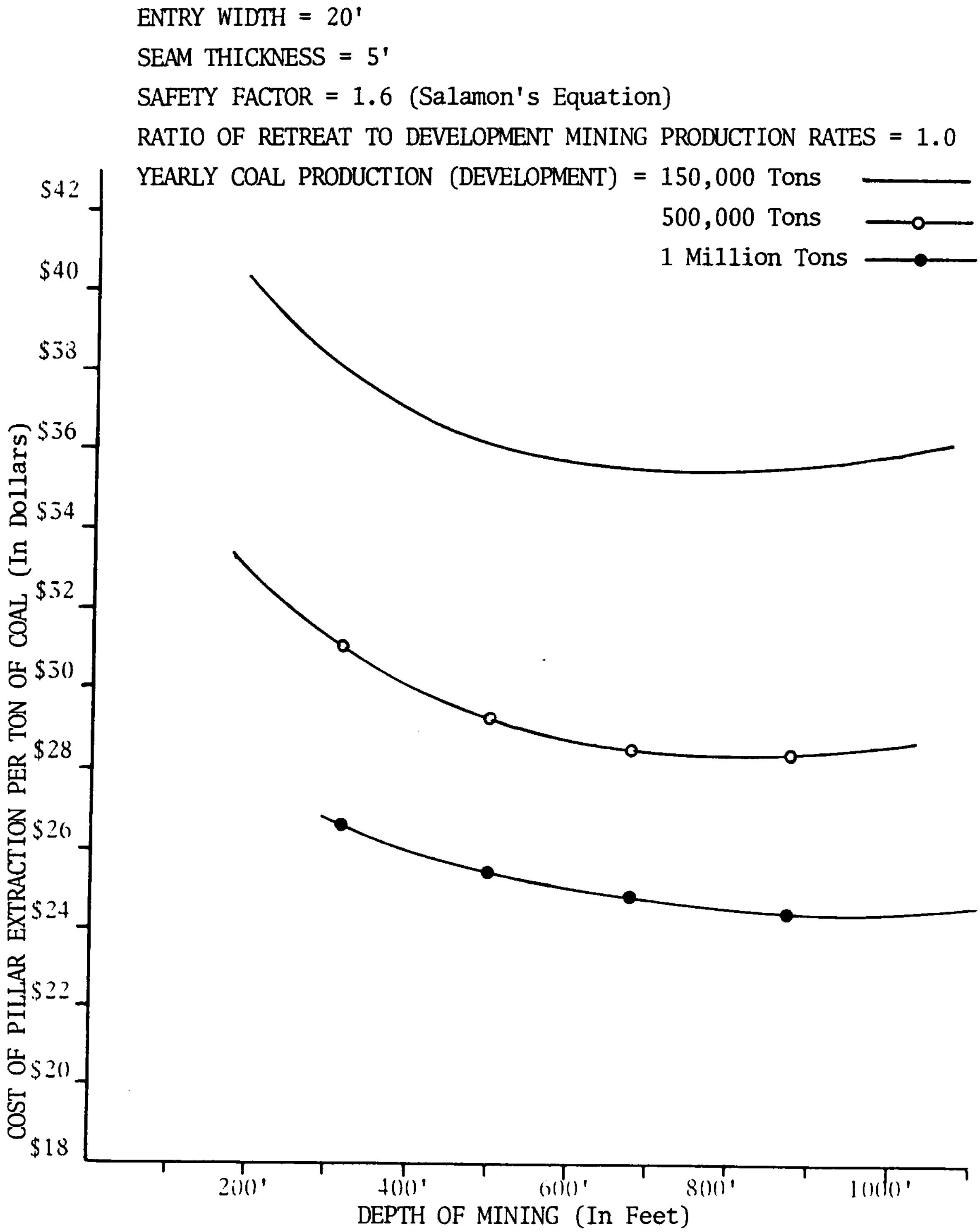
As the coal seam thickness decreases, the cost of pillar extraction per ton of coal increases, as shown in Figure 6.10. It is more profitable to mine thicker seams and get higher productivity than to mine thinner seams.

The area of influence due to subsidence, per ton of coal pillar extraction, decreases with the increasing depth of mining. This is depicted in Figure 6.11. The strain value on the surface due to subsidence also decreases with



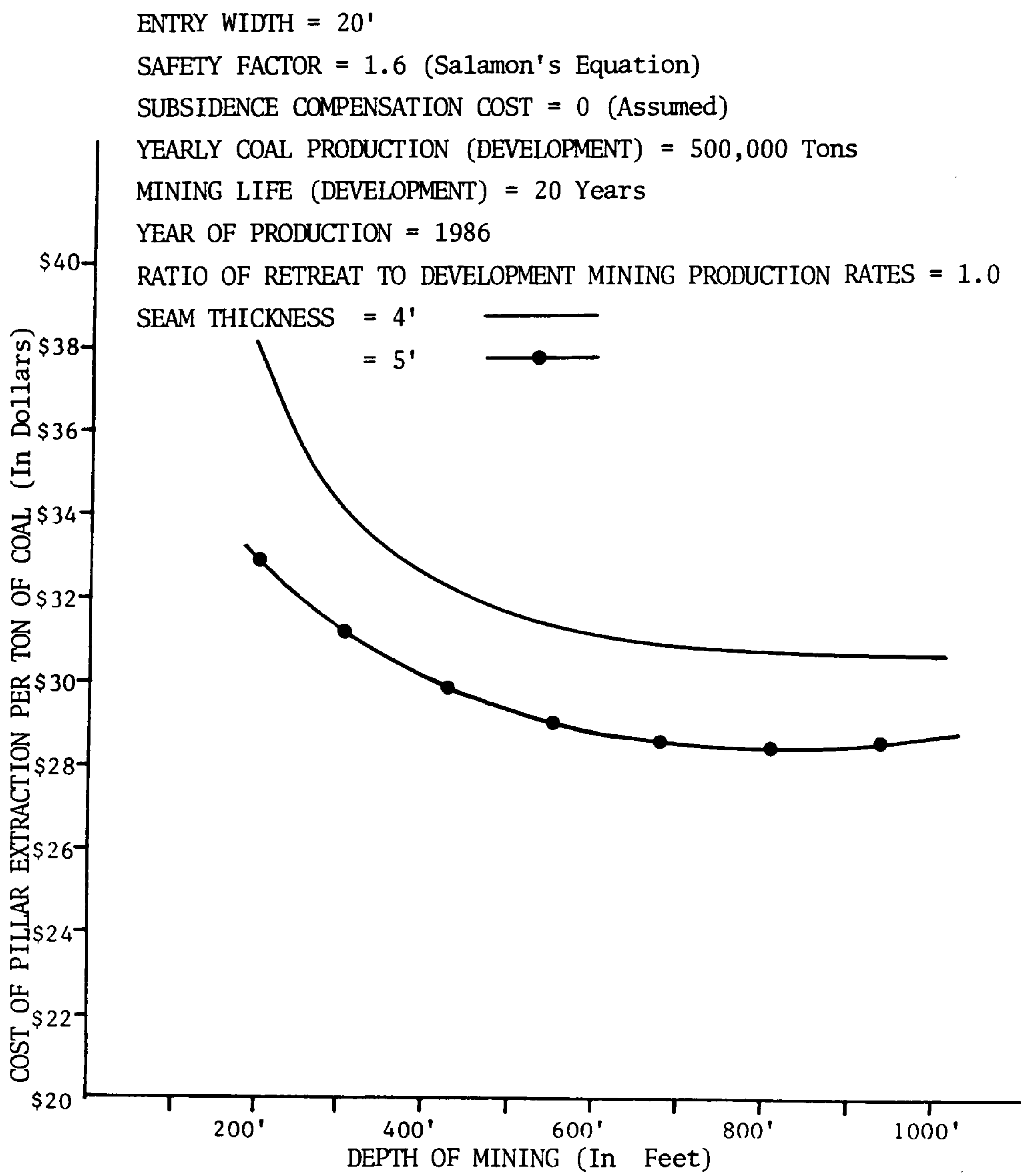
Cost of Pillar Extraction Per Ton of Coal versus Desired Safety Factor for Varying Depths of Mining .

Figure 6.8



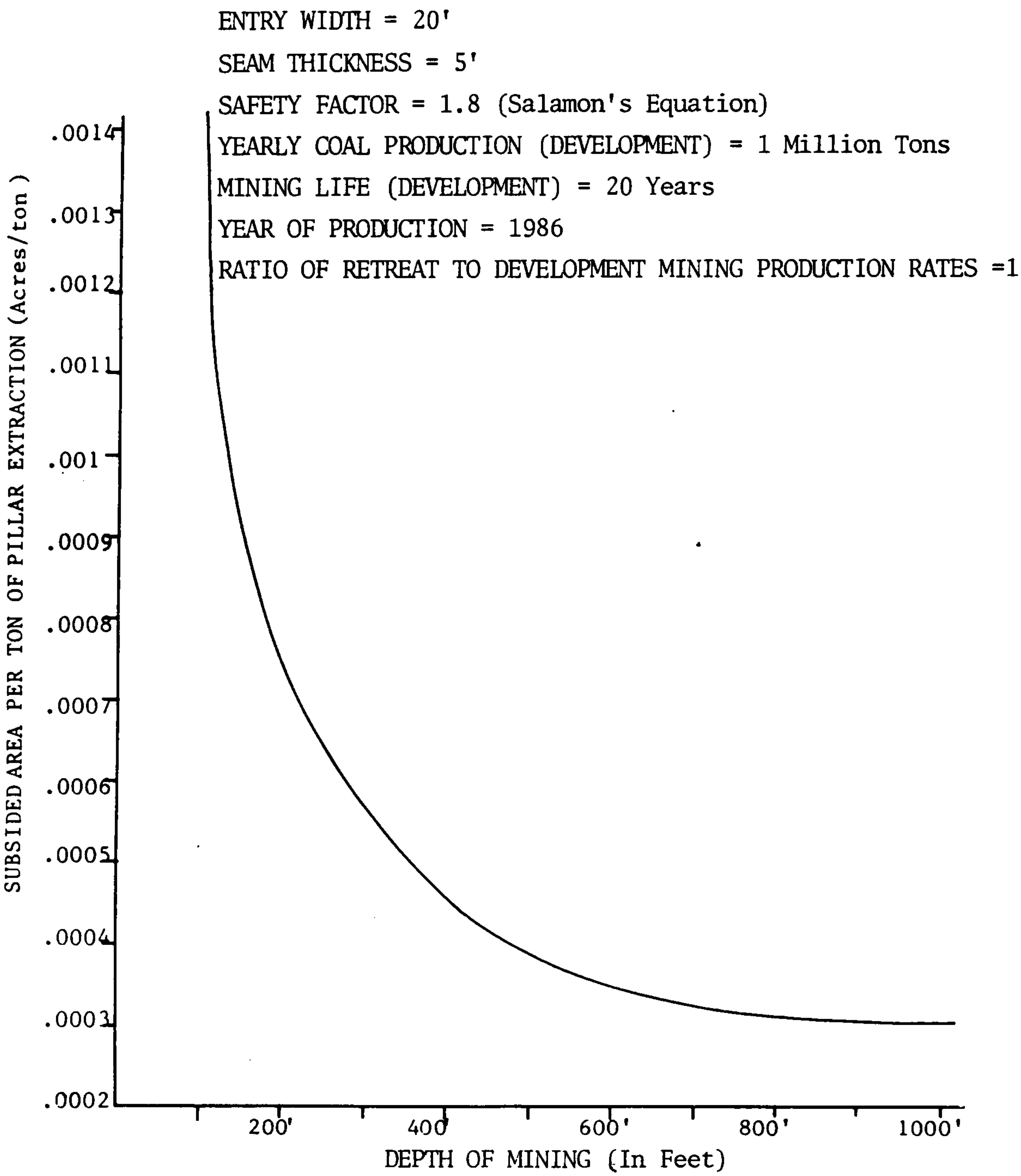
Cost of Pillar Extraction Per Ton of Coal versus Size of Mine for Varying Depths of Mining.

Figure 6.9



Cost of Pillar Extraction Per Ton of Coal versus Coal Seam Thickness for Varying Depths.

Figure 6.10



Subsided Area Per Ton of Pillar Extraction versus Depth of Mining.

Figure 6.11

the increasing depth of the mining operation. The subsidence compensation is computed based on a strain value in conjunction with the area of influence. Hence the subsidence compensation per ton of pillar extraction will decrease with increasing depth.

As the depth of the coal mine increases, it is more likely to encounter coal bump or burst problems. The cost of controlling coal bumps varies with the intensity of bumps and the nature of overburden, roof, floor, and coal. Some coal mines have bump-prone conditions at depths as low as 700 feet, while some mines are bump-free to a depth of 1,500 feet.

Many techniques are available to control coal bumps or bursts. Mine layouts are modified to prevent bumps. Sometimes short holes are drilled into a face, loaded with a predetermined amount of explosive, and shot simultaneously to prevent bumps. In some cases, the roof strata is fractured and the relative stiffness of the loading system increases, thus promoting gradual yielding of the coal rather than dangerous failure. The injection of water under pressure into bump-prone pillars has been practiced on a routine basis for some time in many mines to control coal bumps. These processes of controlling coal bumps require extra investment to mine coal pillars. The cost varies from $0.5 to $5.0 per ton of coal extraction depending on the method of coal bump control and the intensity and nature of the coal bump. The cost figure as obtained from the simulator does not account for coal bump control costs. If the pillar extraction cost is to be computed for coal mines at greater depths, the bump or burst control cost is added to the cost obtained from the simulator output to compute the overall cost of pillar extraction.

If the cost of pillar extraction per ton of coal becomes more than the market price of raw coal, other methods of underground coal extraction such as longwall and shortwall mining techniques, as well as newer, more novel systems are studied and compared to test the economic feasibility of a particular method.

7. CONCLUSIONS

1. A pillar extraction cost model for room-and-pillar coal mining in the United States has been developed.
2. The increase in percentage extraction of coal reserve results in an extension of the life of the mine.
3. The pillar extraction cost model can be used to determine the economic viability of an old, existing, or new mining property or venture. It indicates whether the pillar extraction is economically viable after careful consideration of all engineering, environmental, geologic and other variables pertaining to the pillar extraction project.
4. The model (PILCOST) selects the optimum pillar dimension to minimize the cost of pillar extraction.
5. In the development operation, the cost per ton of coal extraction increases with the increasing depth of mining below the surface.
6. The cost of pillar extraction per acre of mine property increases with the increasing depth of

mining.

7. The cost of pillar extraction per ton of coal decreases with increasing depth up to a specific value, remains nearly constant for further increases in depth, then increases as depth of mining increases. This characterstic is independent of mine size.
8. Where applicable, coal bumps or bursts control cost must be considered and added to the cost figure obtained from the pillar extraction cost model for coal mines operating at greater depths. This will improve the accuracy of cost computation.
9. The cost of pillar extraction per ton of coal decreases with the increasing ratio of pillar extraction to development production rates.
10. The desired safety factor used for basic pillar size design has a significant effect on the cost of pillar extraction per ton of coal.
11. The cost of pillar extraction per ton of coal decreases with the increasing size of the development mining operation or capacity of the mine.
12. As the coal seam thickness decreases, the cost of pillar extraction per ton of coal increases.
13. The area of influence due to subsidence per ton of

coal decreases with increasing depth. The strain value on the surface also decreases due to the increasing depth of mining. If subsidence is based on the area of influence in conjunction with strain value, the compensation per ton of pillar extraction will decrease with the increasing depth.

14. The cost model can be used to determine limiting depth beyond which it is not profitable to mine coal pillars.
15. The cost model provides management information on the relative attractiveness of a pillar extraction project when compared to other alternative methods of underground coal extraction such as longwall and shortwall mining techniques, as well as newer systems.
16. The model can be continually updated in the future to make it as accurate as possible.

Recommendations for Future Research:

1. New cost data should be considered in the future, based on the mine worker's wage contract, annual inflation, the cost of new mine equipment, and the cost of virgin coal property per acre, etc.
2. USBM has recently developed a new mobile roof

support system for retreat mining sections. This research is still in experimental stage. Cost-benefit analysis should be considered to check the economic viability of this new system and the cost model and the computer program should be modified accordingly.

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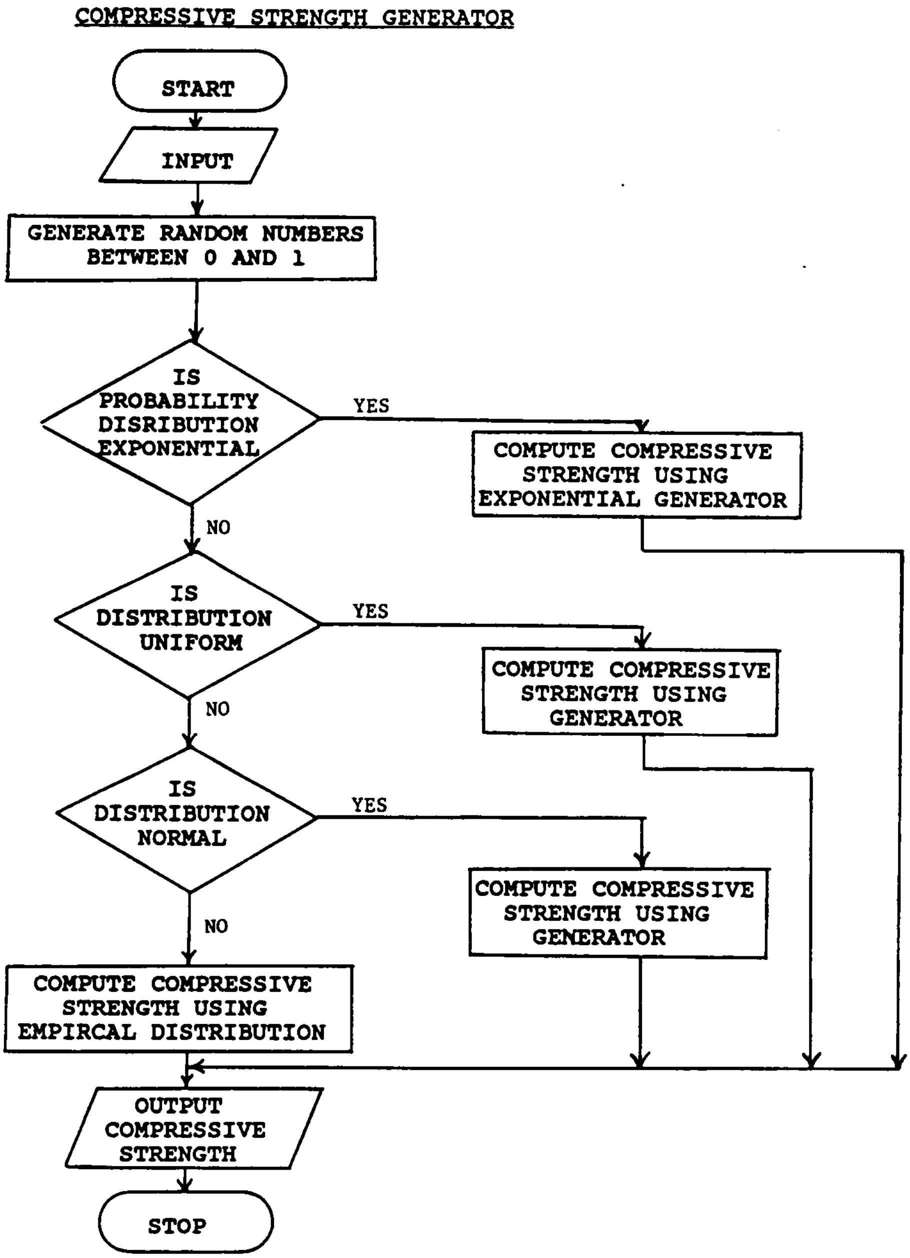
APPENDIX A

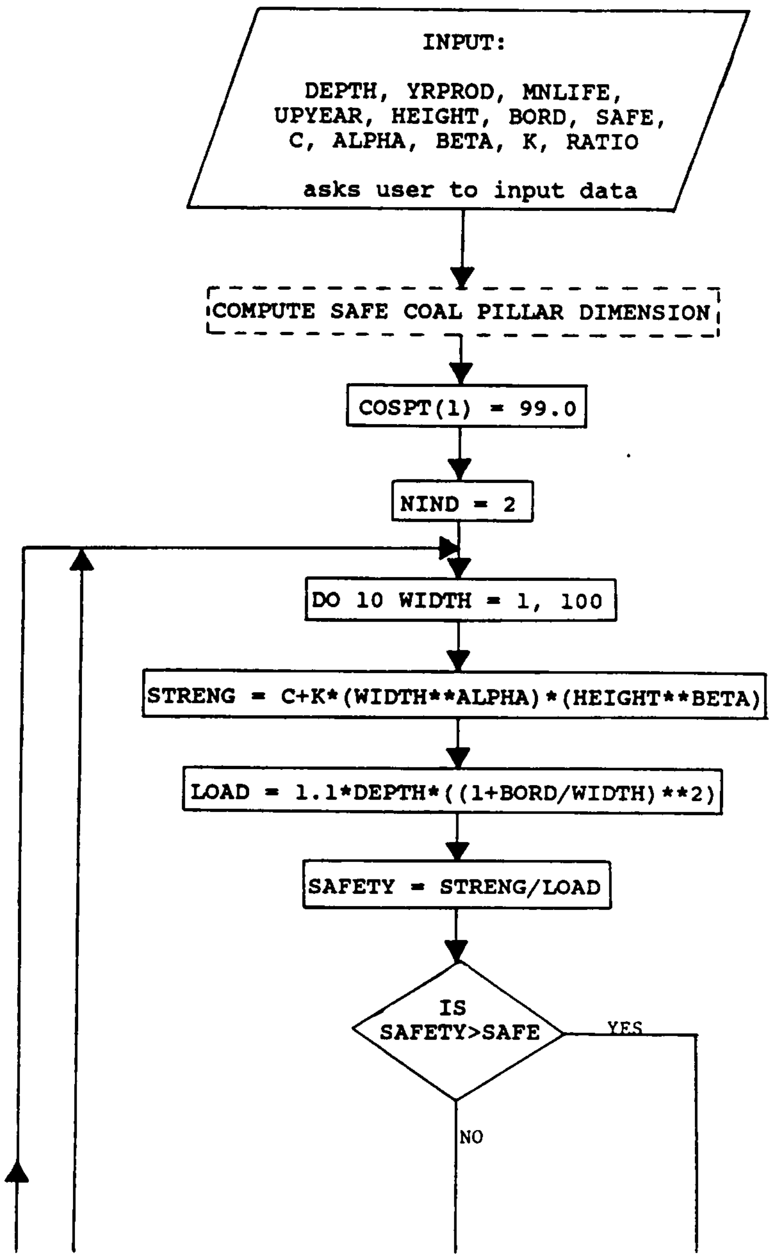
FLOWCHARTS OF THE COMPUTER PROGRAM

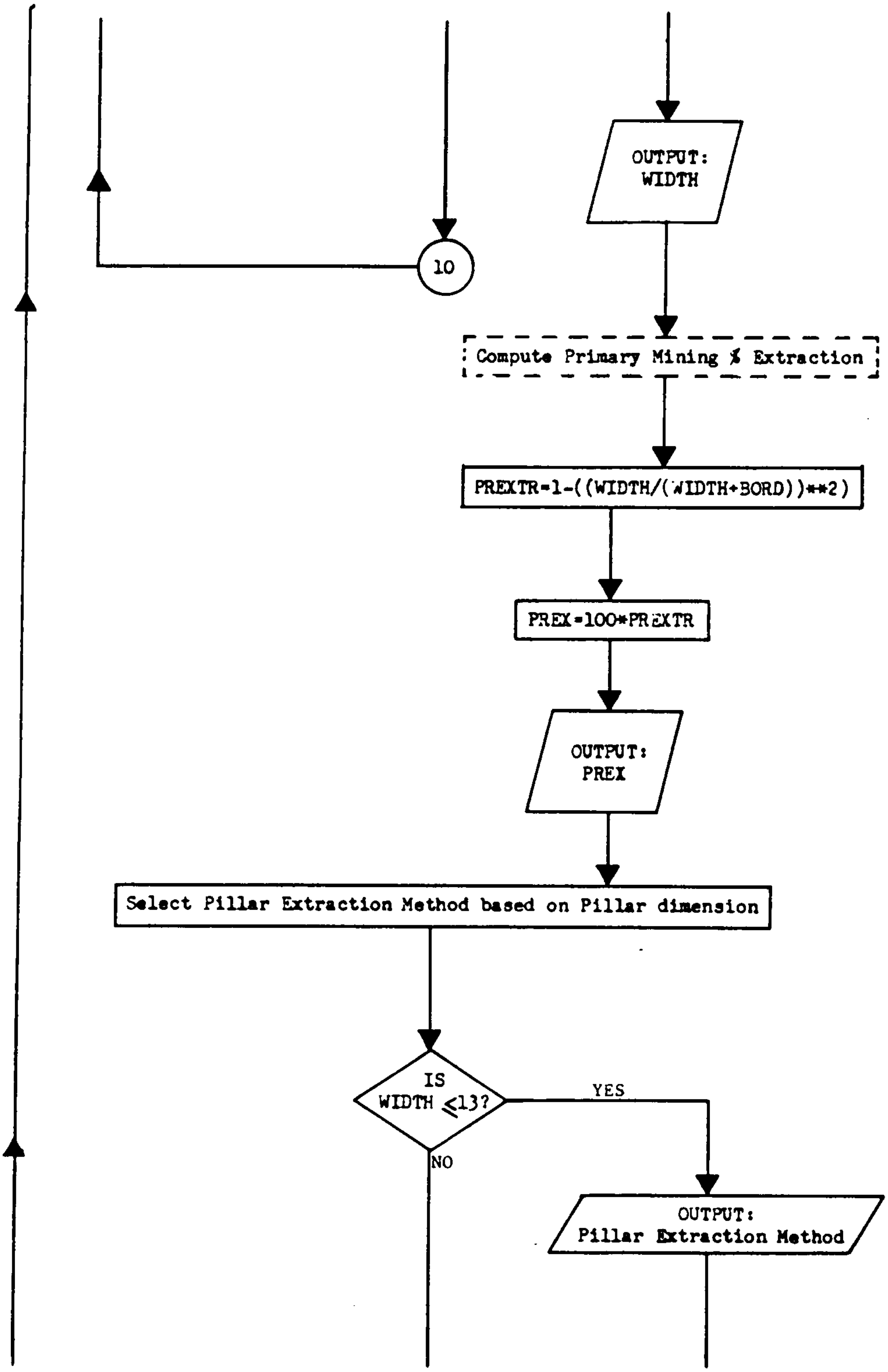


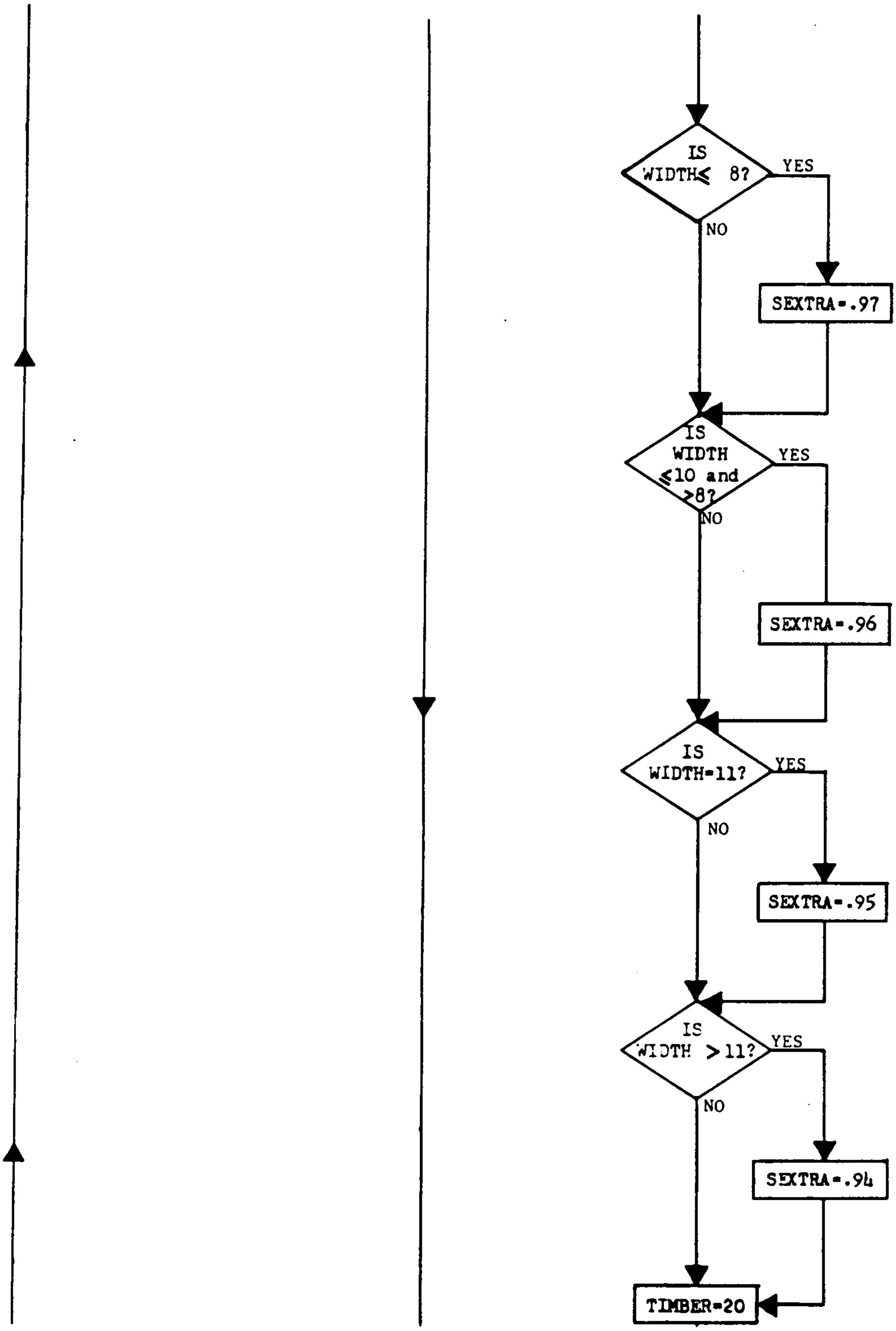
Detail Flowchart of Compressive Strength Generator.

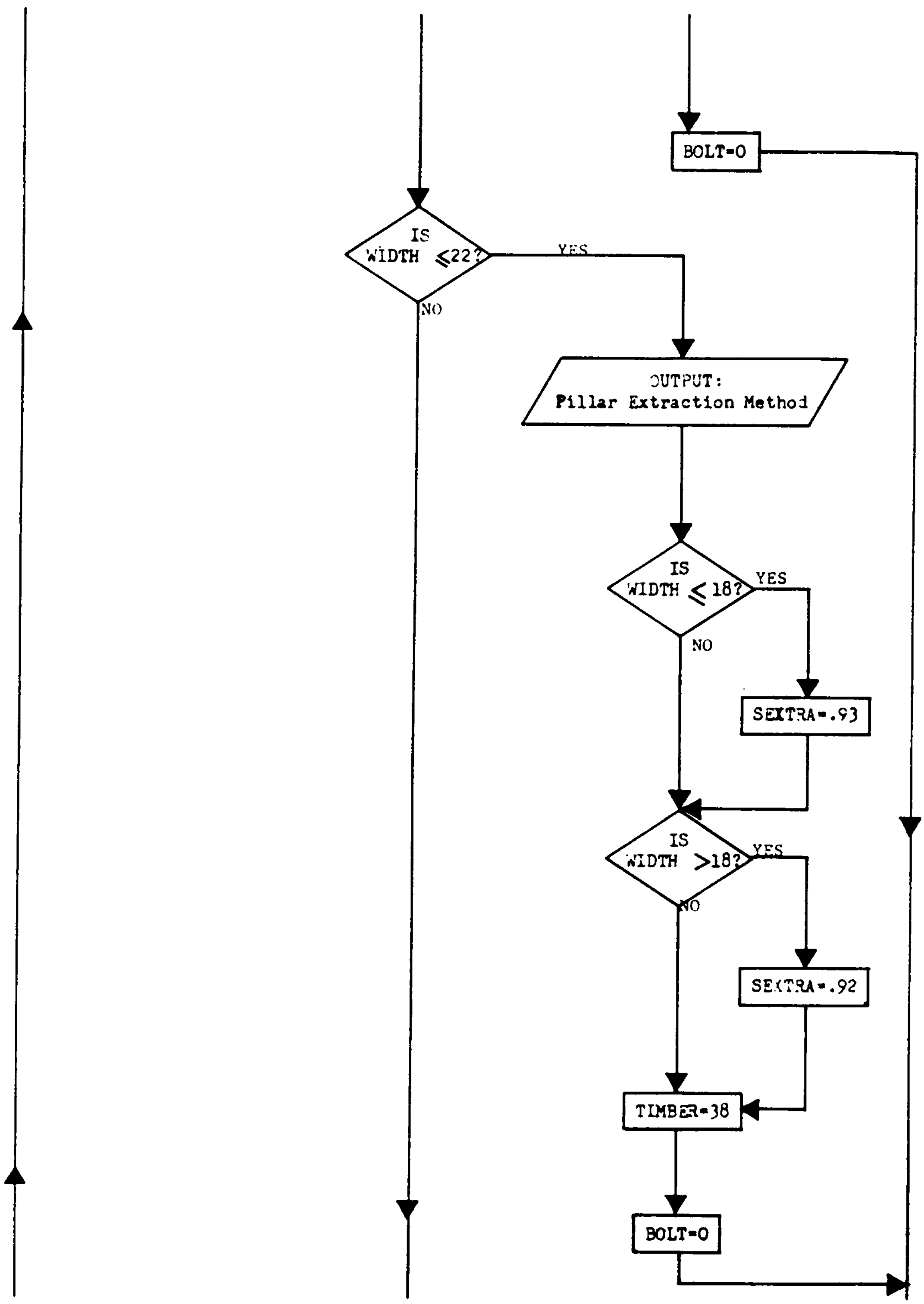
Figure A.1

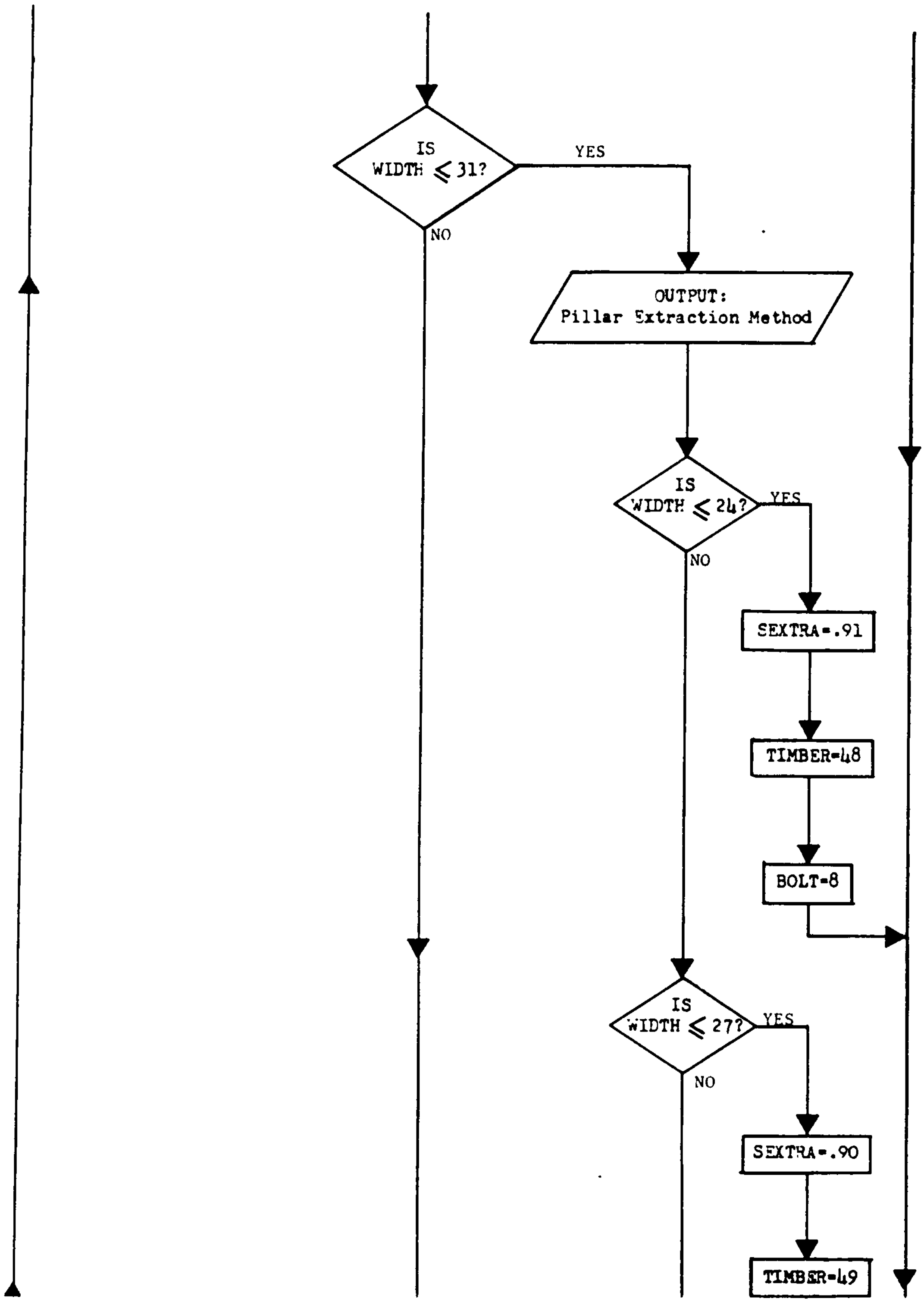
FLOWCHART FOR PILLAR EXTRACTION COST SIMULATOR:

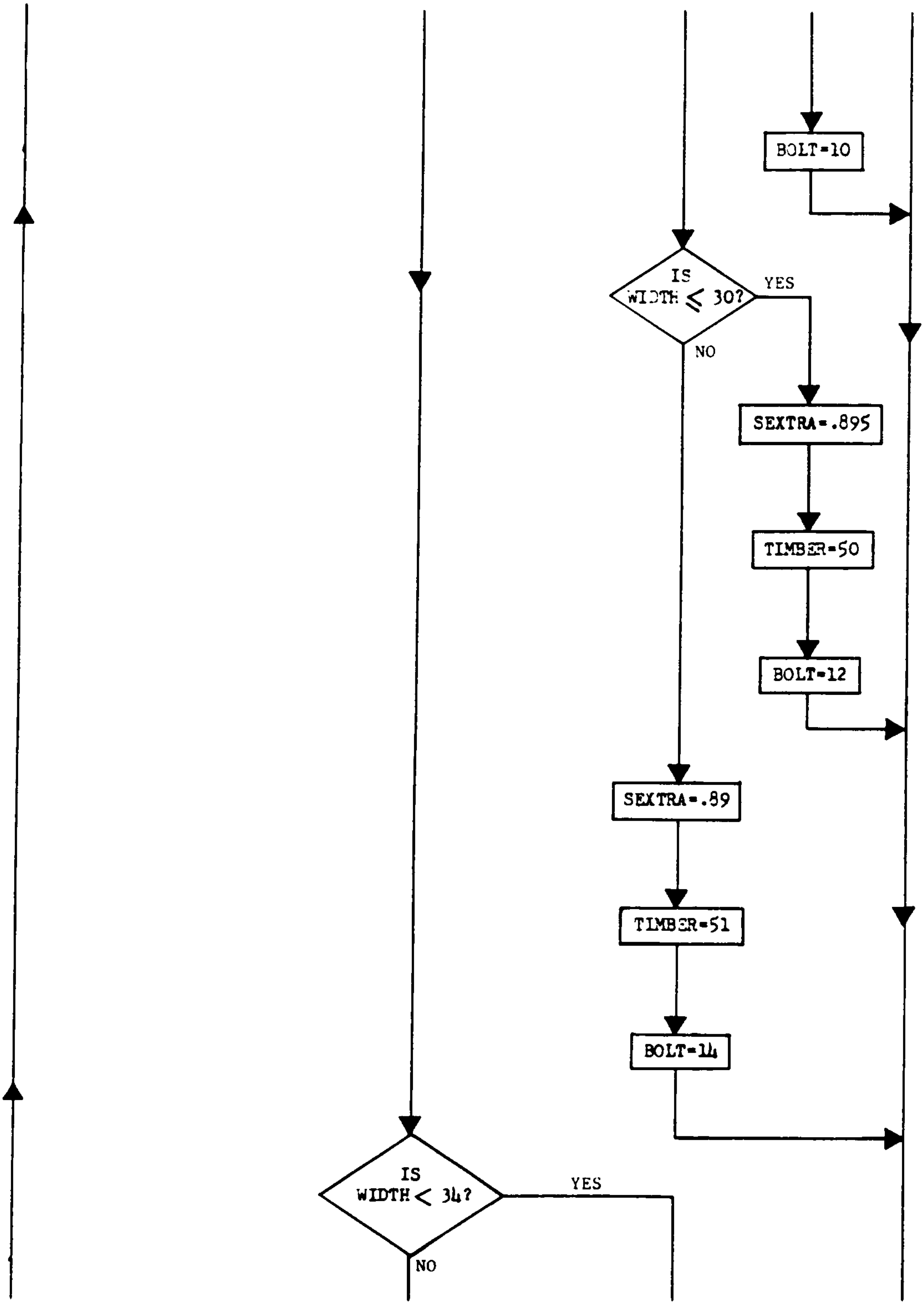


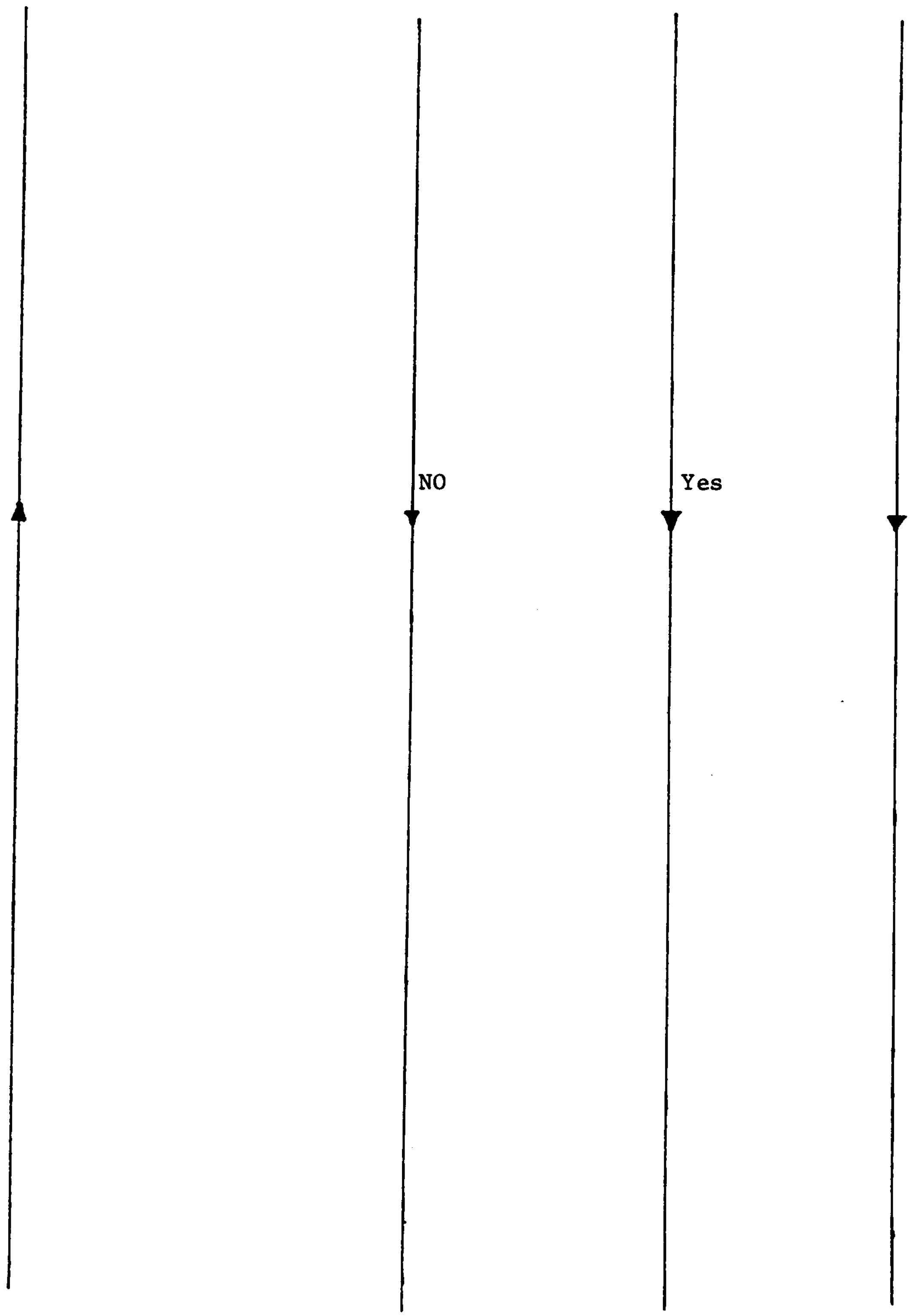

OUTPUT:
WIDTH
10
Compute Primary Mining % Extraction
PREXTR=1-((WIDTH/(WIDTH+BORD))**2)
PREX=100*PREXTR
OUTPUT:
PREX
Select Pillar Extraction Method based on Pillar dimension
IS
WIDTH ≤13?
YES
NO
OUTPUT:
Pillar Extraction Method

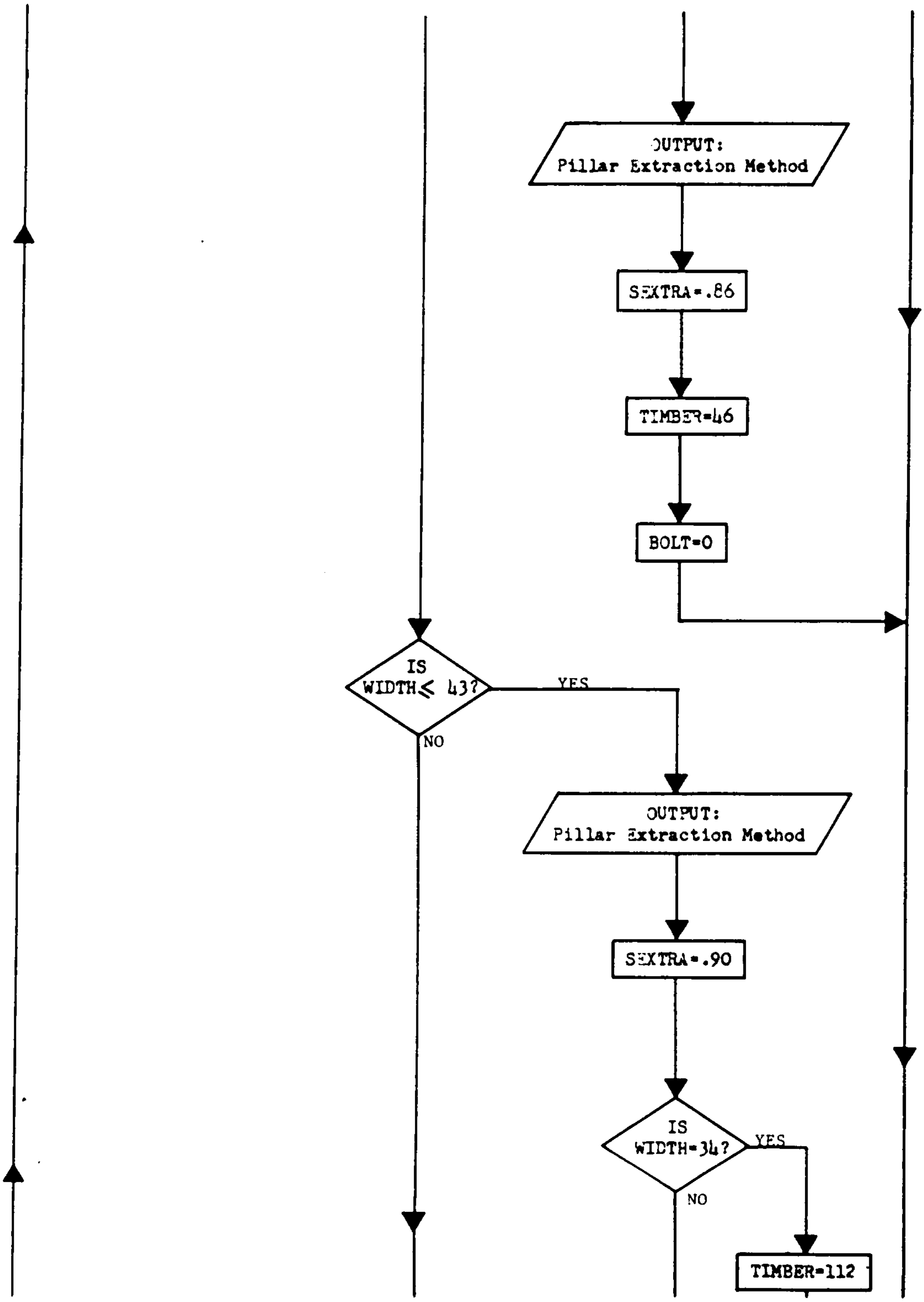

IS
WIDTH≤ 8?
YES
NO
SEXTRA=.97
IS
WIDTH
≤10 and
>8?
YES
NO
SEXTRA=.96
IS
WIDTH=11?
YES
NO
SEXTRA=.95
IS
WIDTH >11?
YES
NO
SEXTRA=.94
TIMBER=20

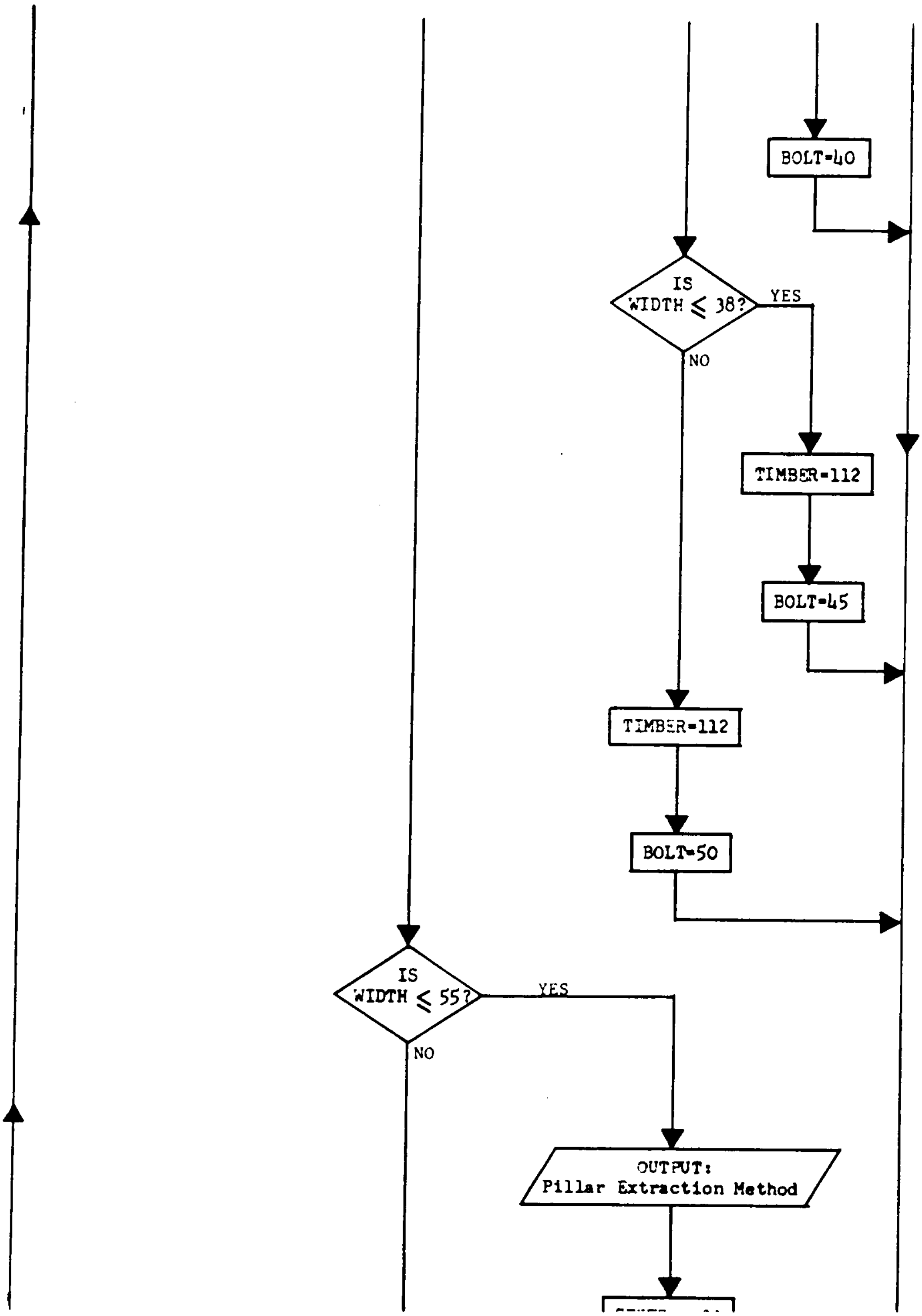

BOLT=0
IS WIDTH ≤22?
YES
NO
OUTPUT: Pillar Extraction Method
IS WIDTH ≤ 18?
YES
NO
SEXTRA=.93
IS WIDTH >18?
YES
NO
SEXTRA=.92
TIMBER=38
BOLT=0

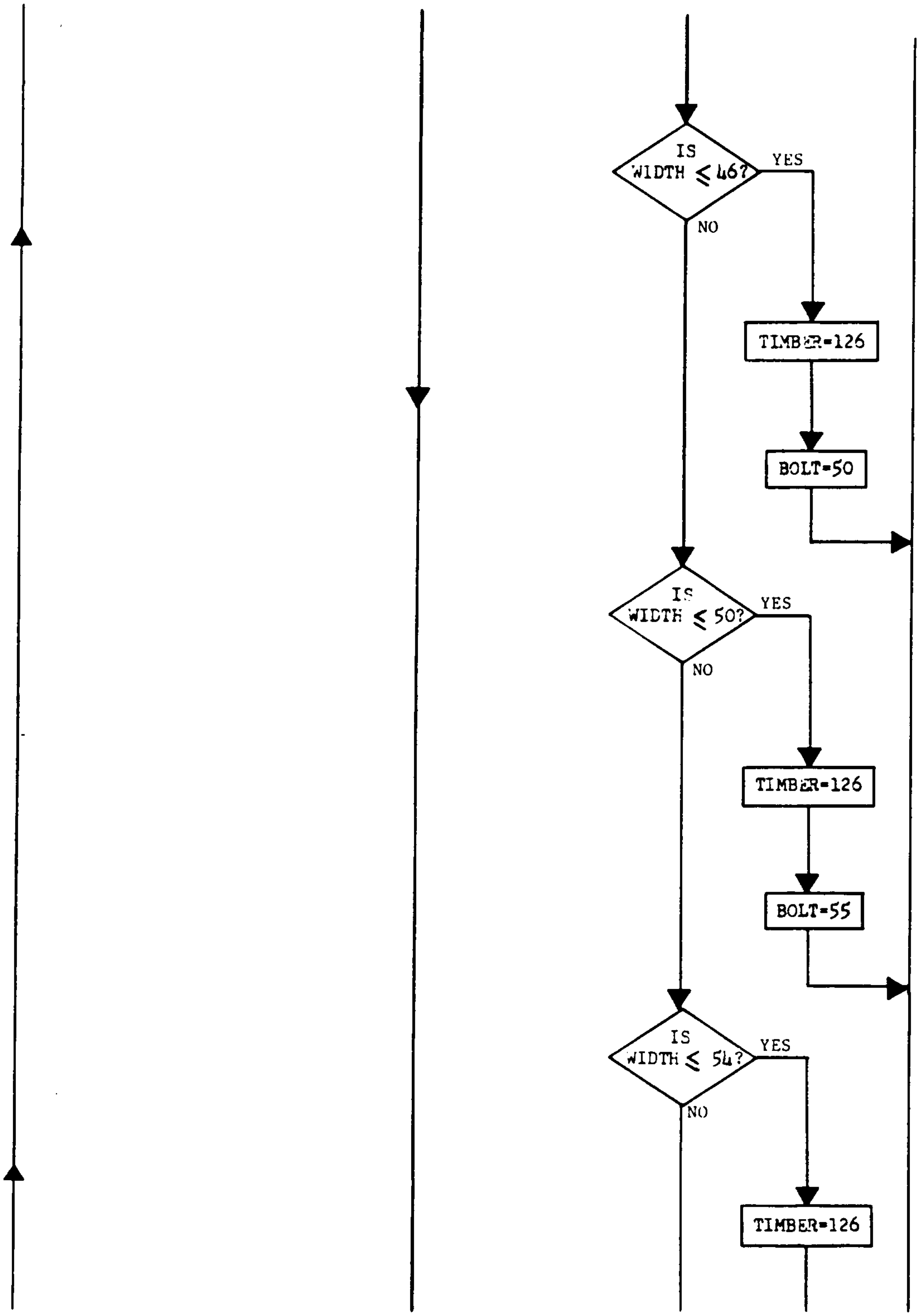

IS
WIDTH ≤ 31?
YES
NO
OUTPUT:
Pillar Extraction Method
IS
WIDTH ≤ 24?
YES
NO
SEXTRA=.91
TIMBER=48
BOLT=8
IS
WIDTH ≤ 27?
YES
NO
SEXTRA=.90
TIMBER=49

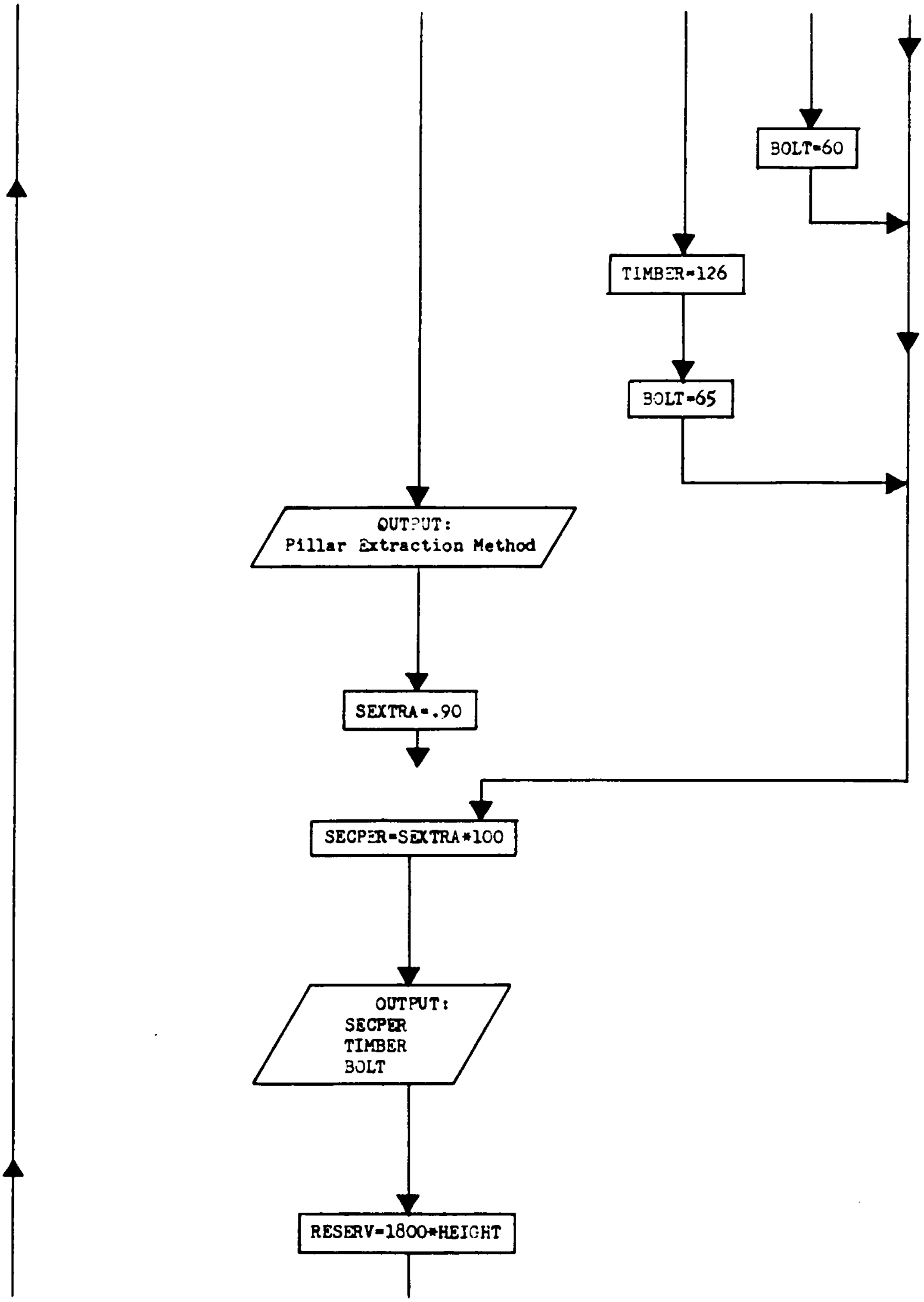

BOLT=10
IS WIDTH ≤ 30?
YES
NO
SEXTRA=.895
TIMBER=50
BOLT=12
SEXTRA=.89
TIMBER=51
BOLT=14
IS WIDTH < 34?
YES
NO

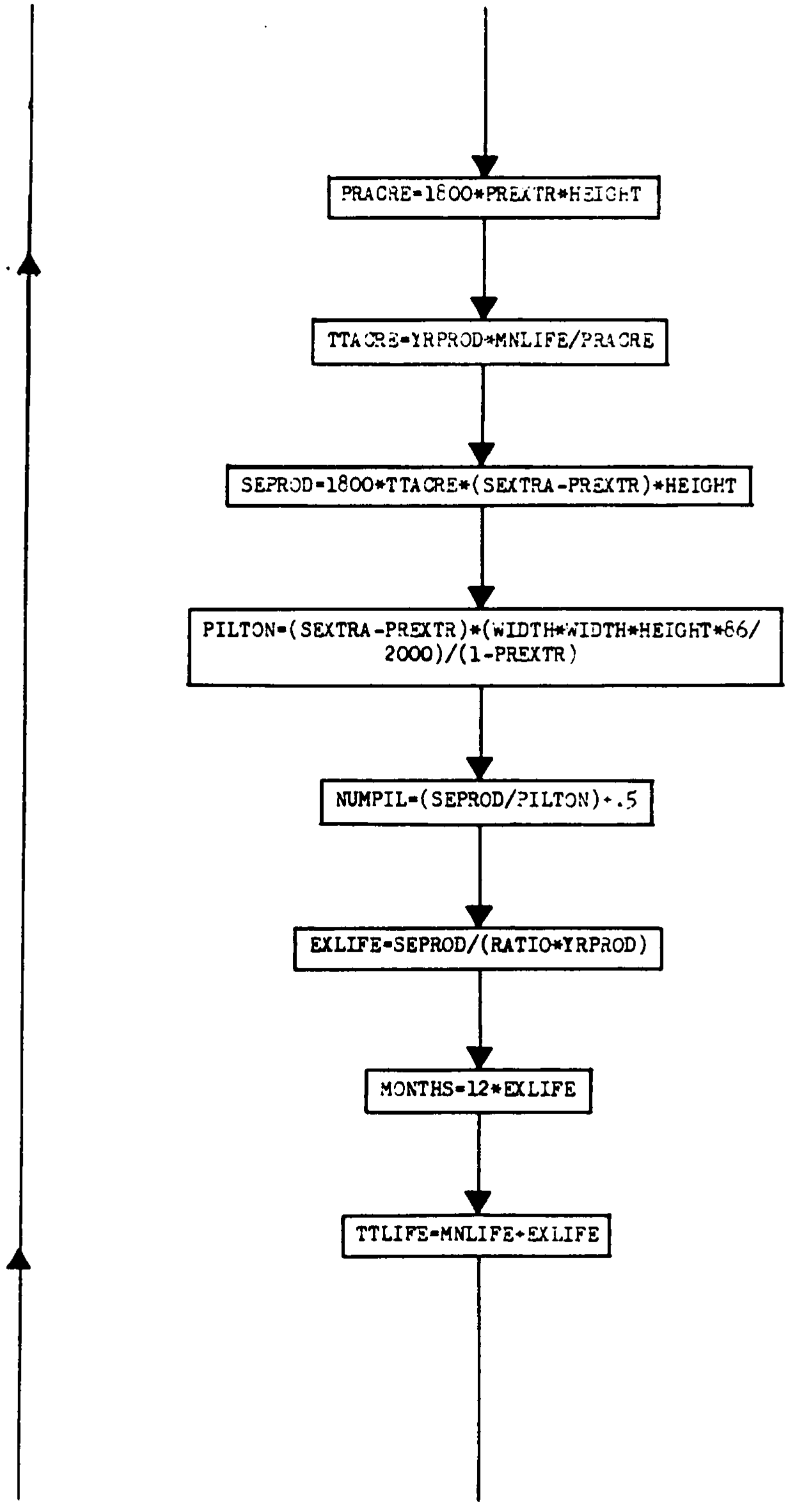

NO
Yes

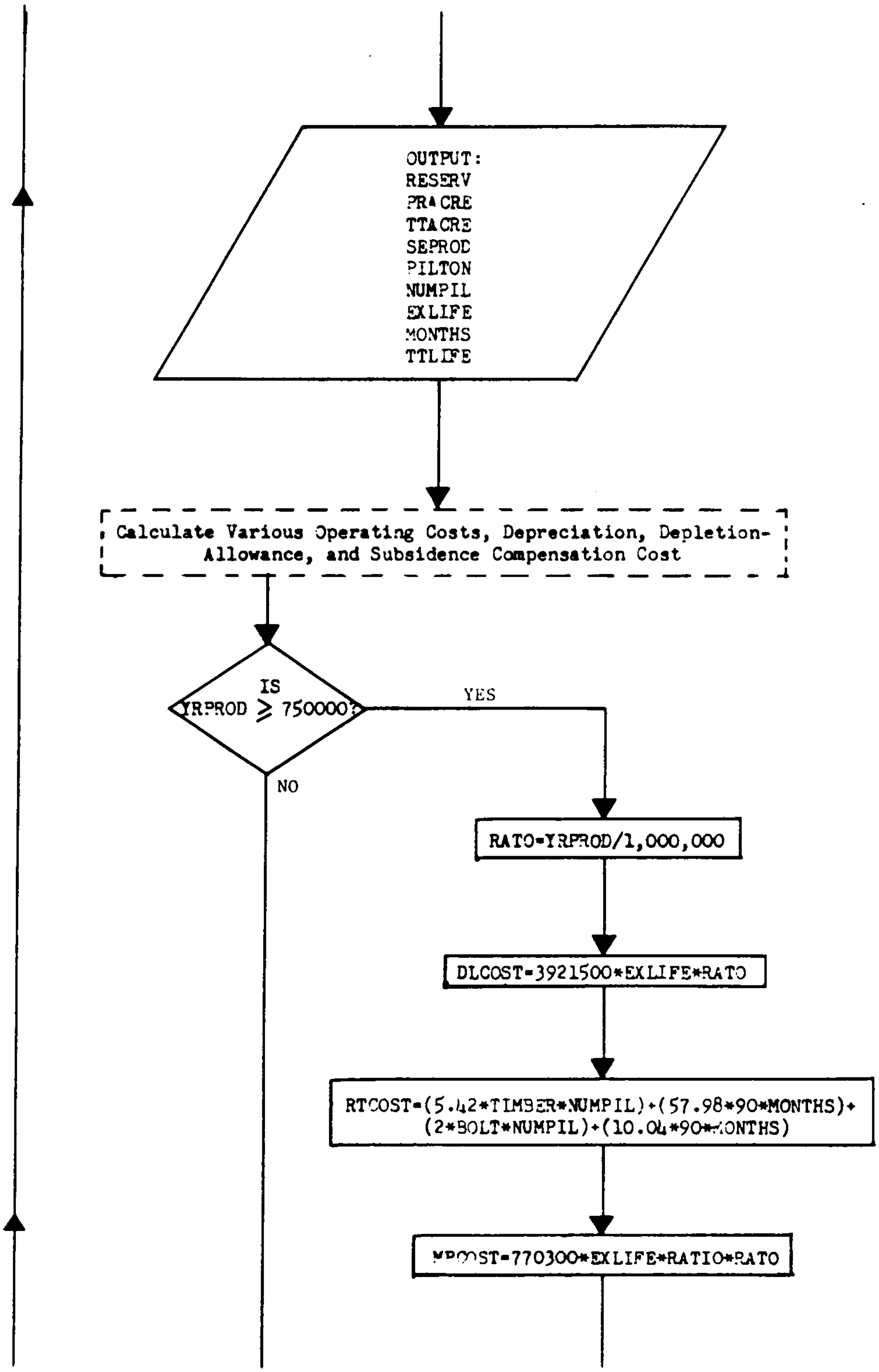

OUTPUT:
Pillar Extraction Method
SEXTRA=.86
TIMBER=46
BOLT=0
IS
WIDTH≤ 43?
YES
NO
OUTPUT:
Pillar Extraction Method
SEXTRA=.90
IS
WIDTH=34?
YES
NO
TIMBER=112

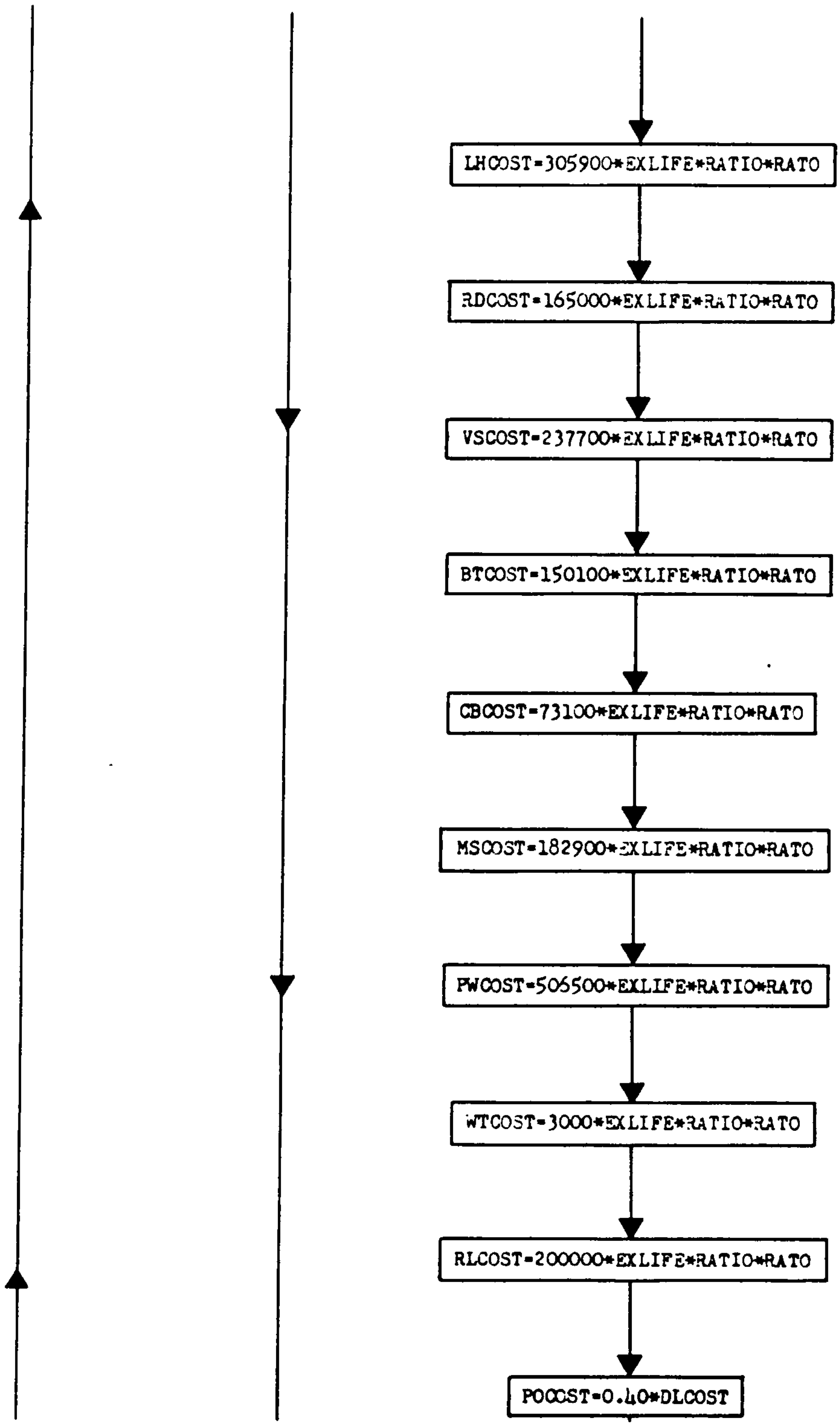

BOLT=40
IS WIDTH ≤ 38?
YES
NO
TIMBER=112
BOLT=45
TIMBER=112
BOLT=50
IS WIDTH ≤ 55?
YES
NO
OUTPUT:
Pillar Extraction Method

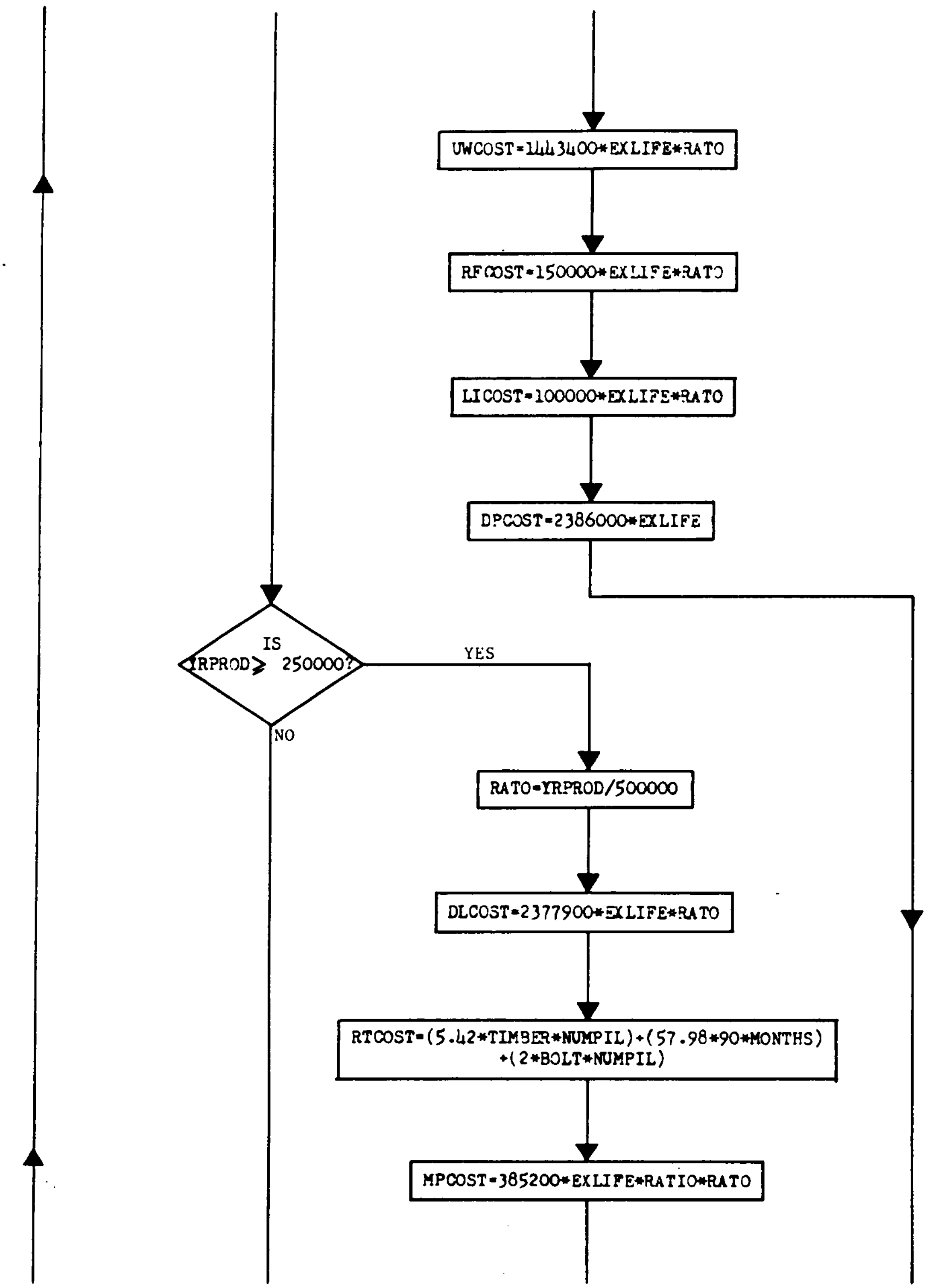

IS WIDTH ≤ 46?
YES
NO
TIMBER=126
BOLT=50
IS WIDTH ≤ 50?
YES
NO
TIMBER=126
BOLT=55
IS WIDTH ≤ 54?
YES
NO
TIMBER=126

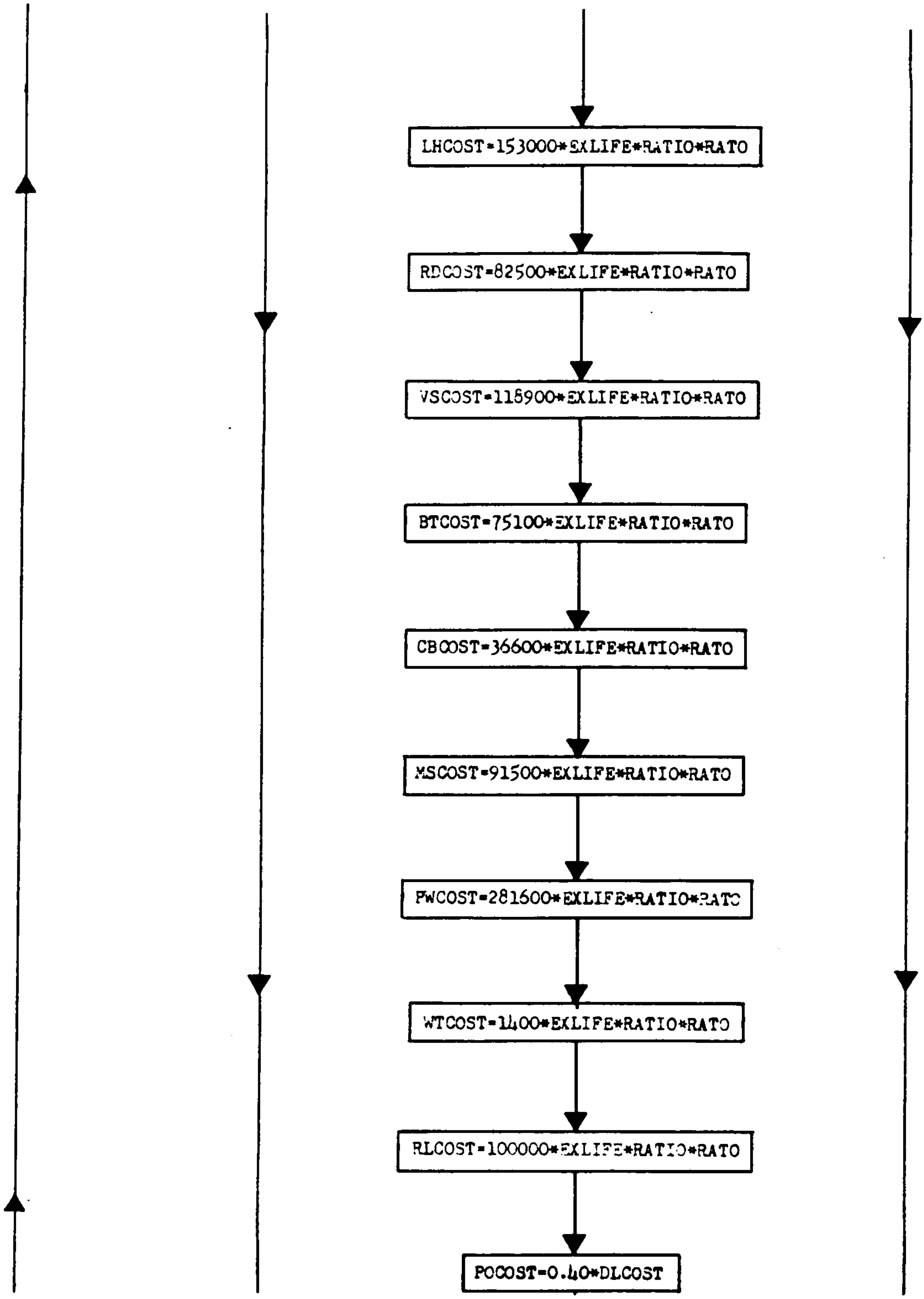

BOLT=60
TIMBER=126
BOLT=65
OUTPUT:
Pillar Extraction Method
SEXTRA=.90
SECPER=SEXTRA*100
OUTPUT:
SECPER
TIMBER
BOLT
RESERV=1800*HEIGHT

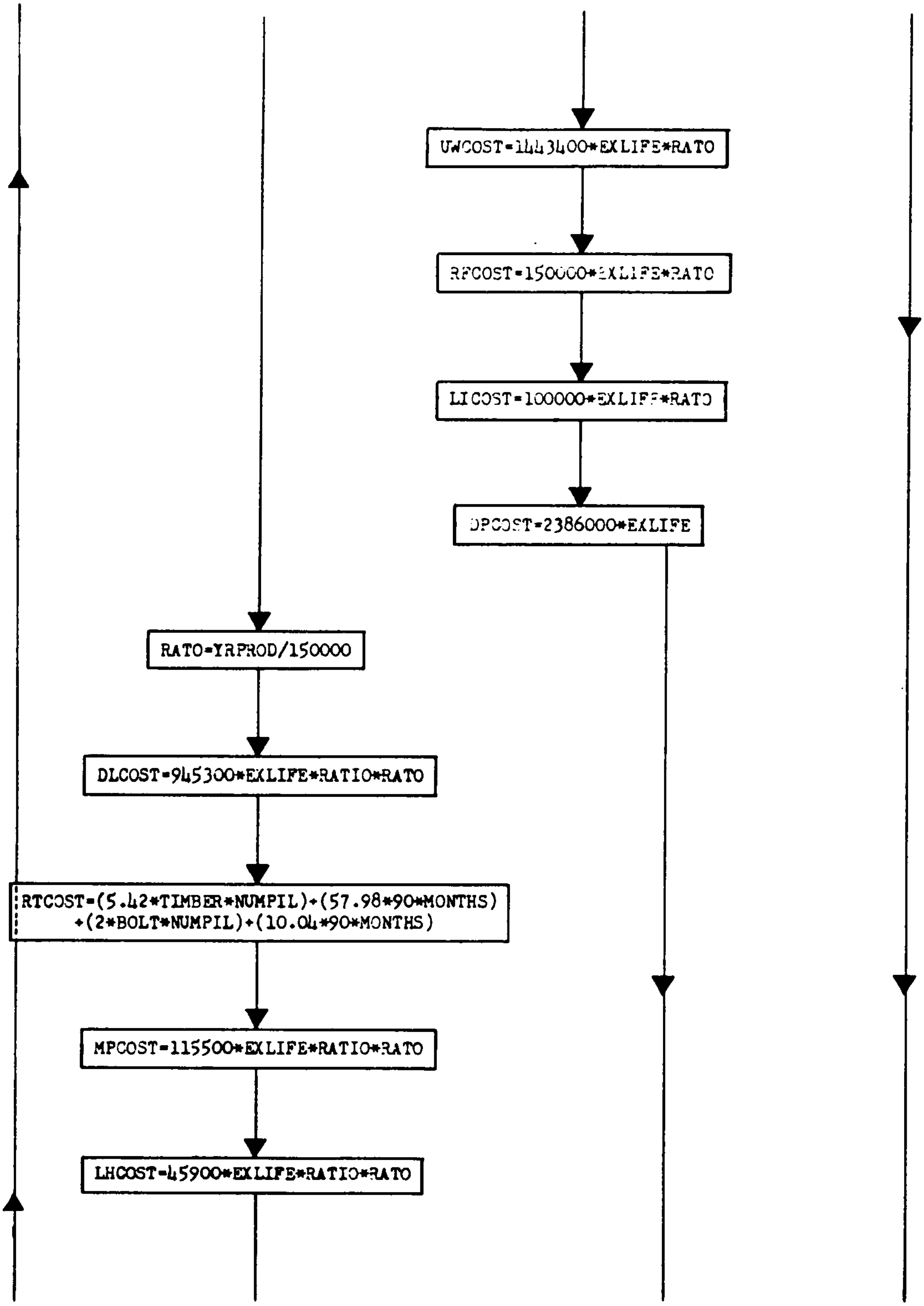

PRACRE=1800*PREXTR*HEIGHT
TTACRE=YRPROD*MNLIFE/PRACRE
SEPROD=1800*TTACRE*(SEXTRA-PREXTR)*HEIGHT
PILTON=(SEXTRA-PREXTR)*(WIDTH*WIDTH*HEIGHT*66/2000)/(1-PREXTR)
NUMPIL=(SEPROD/PILTON)+.5
EXLIFE=SEPROD/(RATIO*YRPROD)
MONTHS=12*EXLIFE
TTLIFE=MNLIFE+EXLIFE

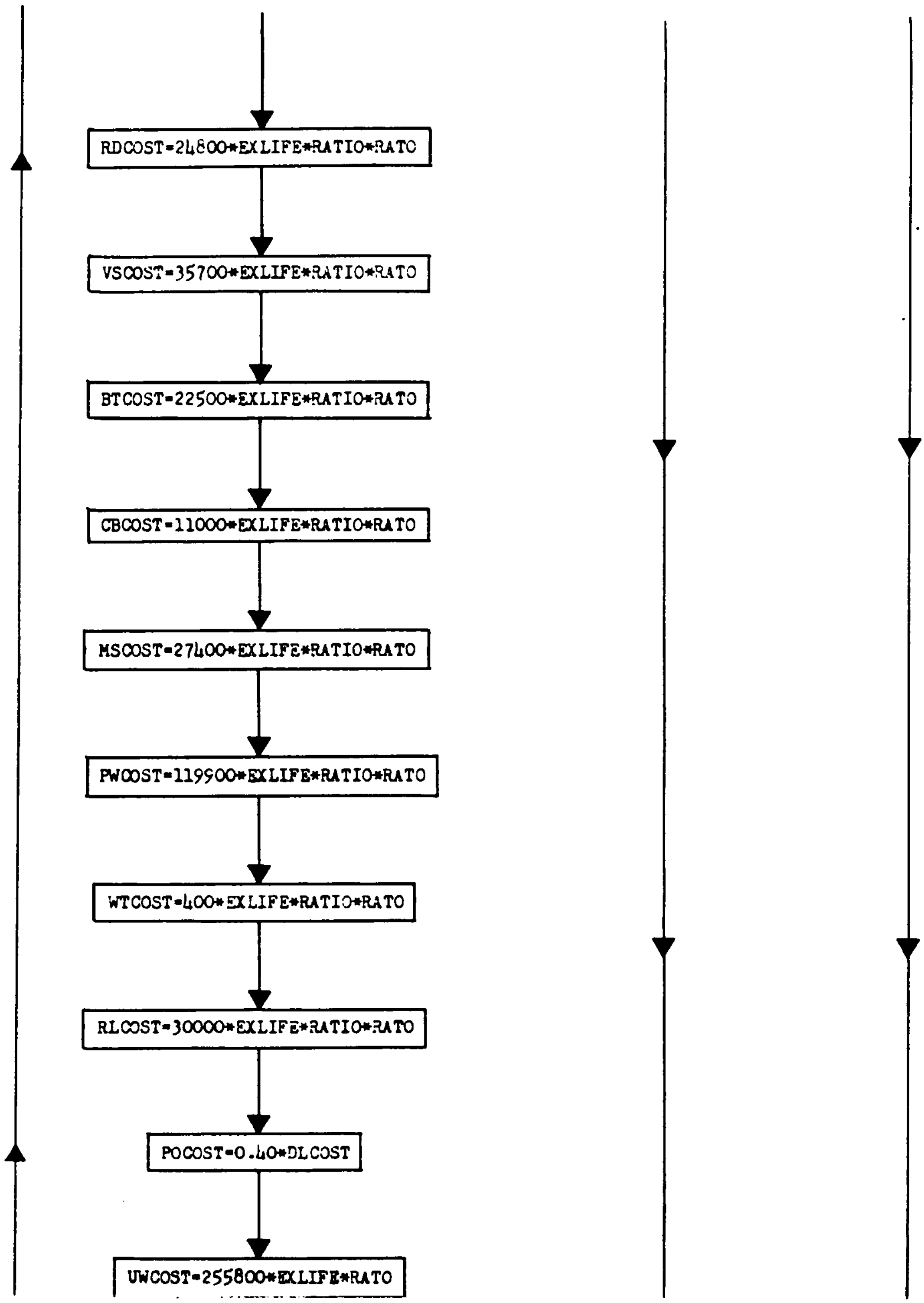

OUTPUT:
RESERV
PRACRE
TTACRE
SEPROD
PILTON
NUMPIL
EXLIFE
MONTHS
TTLIFE
Calculate Various Operating Costs, Depreciation, Depletion-Allowance, and Subsidence Compensation Cost
IS YRPROD ≥ 750000?
YES
NO
RATO=YRPROD/1,000,000
DLCOST=3921500*EXLIFE*RATO
RTCOST=(5.42*TIMBER*NUMPIL)+(57.98*90*MONTHS)+(2*BOLT*NUMPIL)+(10.04*90*MONTHS)
ST=770300*EXLIFE*RATIO*RATO

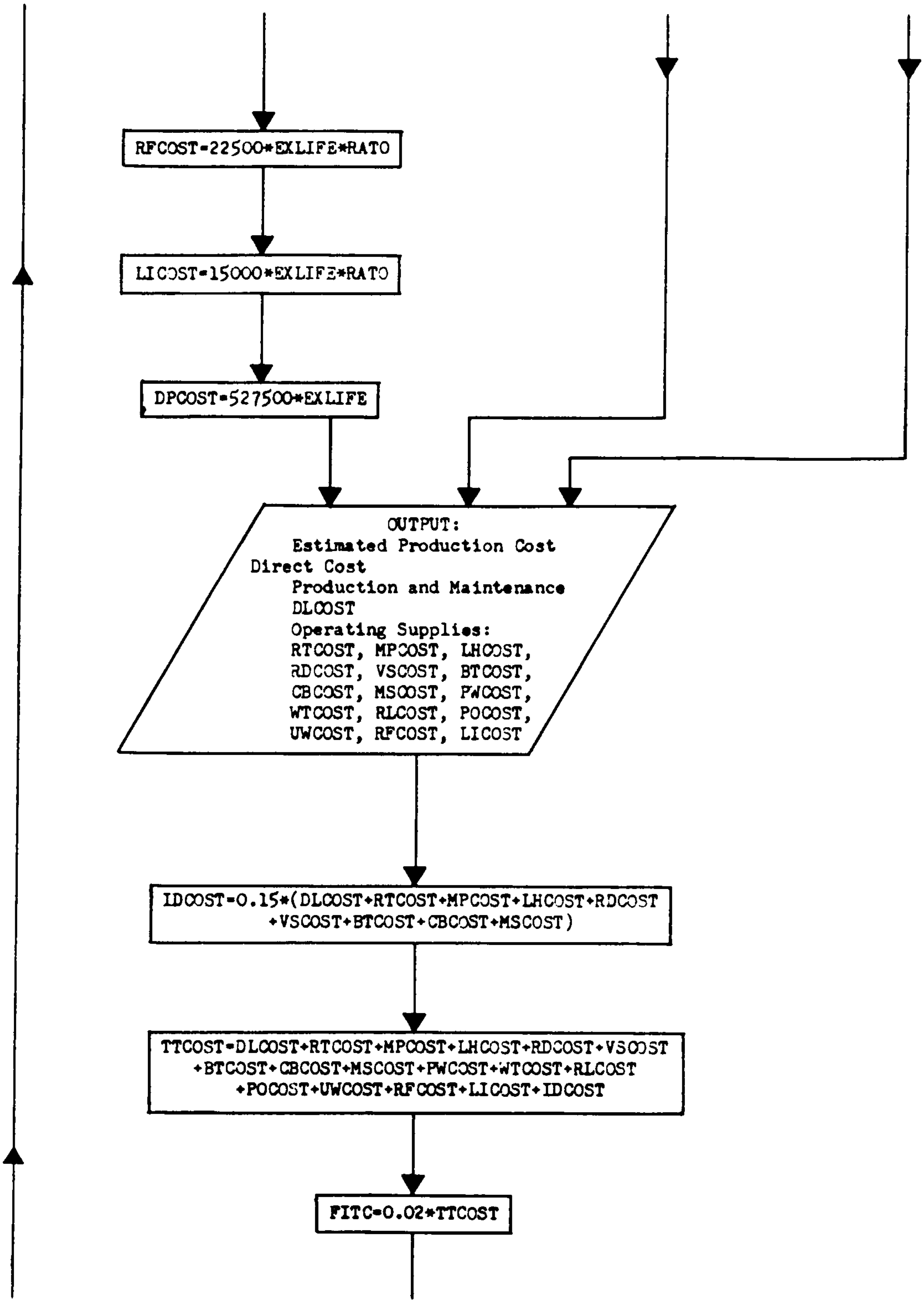

LHCOST=305900*EXLIFE*RATIO*RATO
RDCOST=165000*EXLIFE*RATIO*RATO
VSCOST=237700*EXLIFE*RATIO*RATO
BTCOST=150100*EXLIFE*RATIO*RATO
CBCOST=73100*EXLIFE*RATIO*RATO
MSCOST=182900*EXLIFE*RATIO*RATO
PWCOST=506500*EXLIFE*RATIO*RATO
WTCOST=3000*EXLIFE*RATIO*RATO
RLCOST=200000*EXLIFE*RATIO*RATO
POCOST=0.40*DLCOST

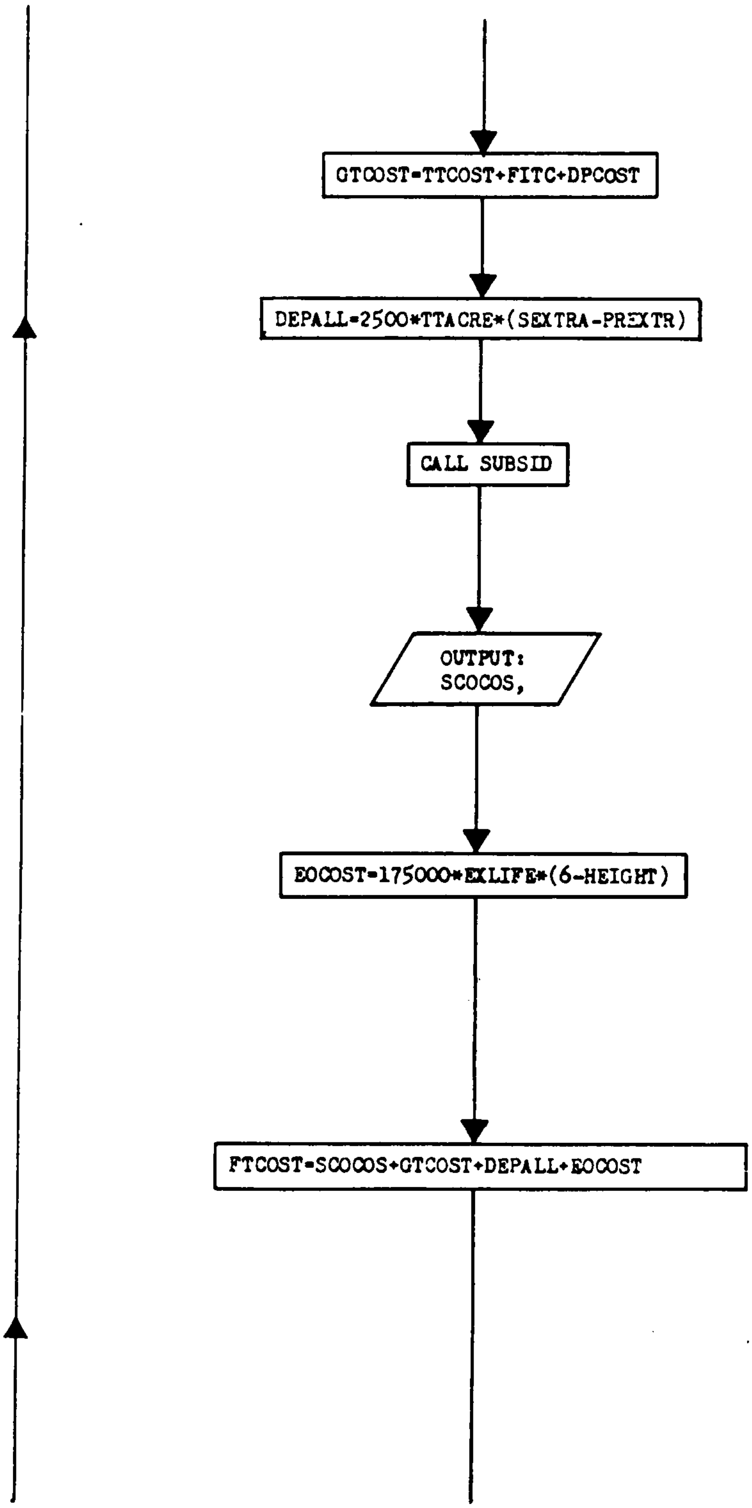

UWCOST=1443400*EXLIFE*RATO
RFCOST=150000*EXLIFE*RATO
LICOST=100000*EXLIFE*RATO
DPCOST=2386000*EXLIFE
IS YRPROD ≥ 250000?
YES
NO
RATO=YRPROD/500000
DLCOST=2377900*EXLIFE*RATO
RTCOST=(5.42*TIMBER*NUMPIL)+(57.98*90*MONTHS)+(2*BOLT*NUMPIL)
MPCOST=385200*EXLIFE*RATIO*RATO

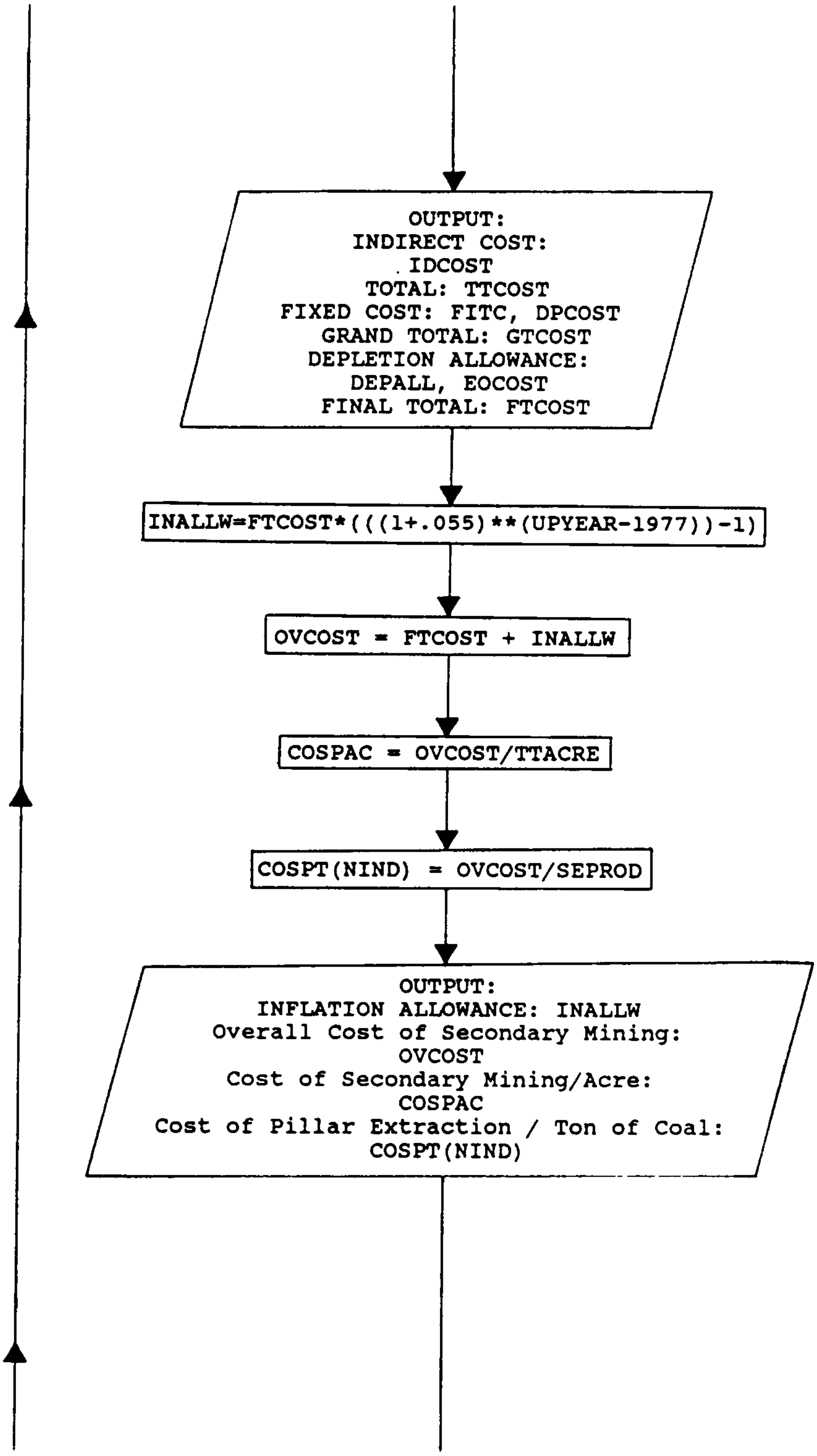

LHCOST=153000*EXLIFE*RATIO*RATO
RDCOST=82500*EXLIFE*RATIO*RATO
VSCOST=118900*EXLIFE*RATIO*RATO
BTCOST=75100*EXLIFE*RATIO*RATO
CBCOST=36600*EXLIFE*RATIO*RATO
MSCOST=91500*EXLIFE*RATIO*RATO
FWCOST=281600*EXLIFE*RATIO*RATO
WTCOST=1400*EXLIFE*RATIO*RATO
RLCOST=100000*EXLIFE*RATIO*RATO
POCOST=0.40*DLCOST

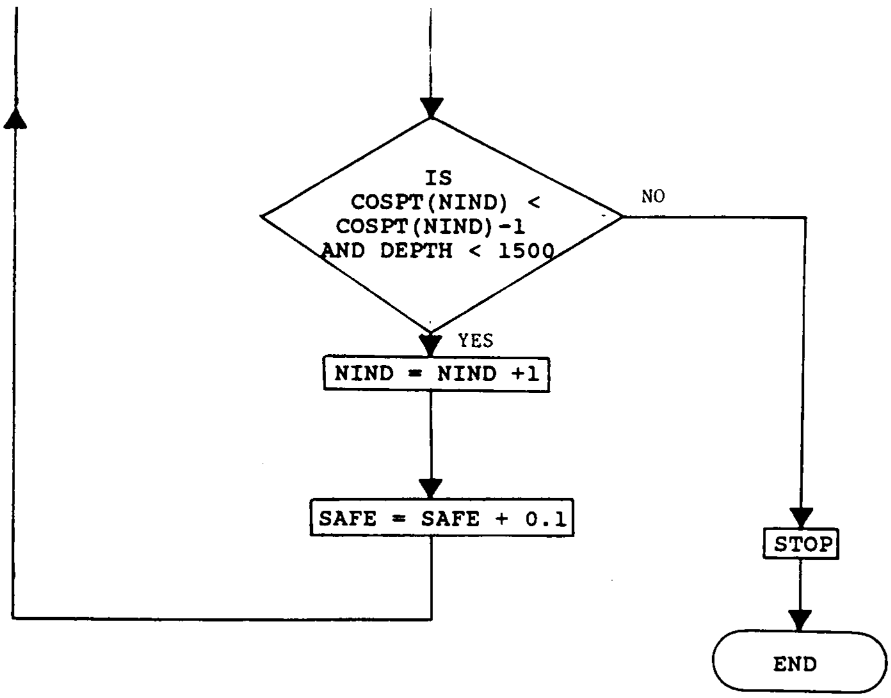

UWCOST=1443400*EXLIFE*RATO
RFCOST=150000*EXLIFE*RATO
LICOST=100000*EXLIFE*RATO
DPCOST=2386000*EXLIFE
RATO=YRPROD/150000
DLCOST=945300*EXLIFE*RATIO*RATO
RTCOST=(5.42*TIMBER*NUMPIL)+(57.98*90*MONTHS)
+(2*BOLT*NUMPIL)+(10.04*90*MONTHS)
MPCOST=115500*EXLIFE*RATIO*RATO
LHCOST=45900*EXLIFE*RATIO*RATO


RDCOST=24800*EXLIFE*RATIO*RATO
VSCOST=35700*EXLIFE*RATIO*RATO
BTCOST=22500*EXLIFE*RATIO*RATO
CBCOST=11000*EXLIFE*RATIO*RATO
MSCOST=27400*EXLIFE*RATIO*RATO
PWCOST=119900*EXLIFE*RATIO*RATO
WTCOST=400*EXLIFE*RATIO*RATO
RLCOST=30000*EXLIFE*RATIO*RATO
POCOST=0.40*DLCOST
UWCOST=255800*EXLIFE*RATO


RFCOST=22500*EXLIFE*RATO
LICOST=15000*EXLIFE*RATO
DPCOST=527500*EXLIFE
OUTPUT:
Estimated Production Cost
Direct Cost
Production and Maintenance
DLCOST
Operating Supplies:
RTCOST, MPCOST, LHCOST,
RDCOST, VSCOST, BTCOST,
CBCOST, MSCOST, PWCOST,
WTCOST, RLCOST, POCOST,
UWCOST, RFCOST, LICOST
IDCOST=0.15*(DLCOST+RTCOST+MPCOST+LHCOST+RDCOST
+VSCOST+BTCOST+CBCOST+MSCOST)
TTCOST=DLCOST+RTCOST+MPCOST+LHCOST+RDCOST+VSCOST
+BTCOST+CBCOST+MSCOST+PWCOST+WTCOST+RLCOST
+POCOST+UWCOST+RFCOST+LICOST+IDCOST
FITC=0.02*TTCOST


GTCOST=TTCOST+FITC+DPCOST
DEPALL=2500*TTACRE*(SEXTRA-PREXTR)
CALL SUBSID
OUTPUT:
SCOCOS,
EOCOST=175000*EXLIFE*(6-HEIGHT)
FTCOST=SCOCOS+GTCOST+DEPALL+EOCOST

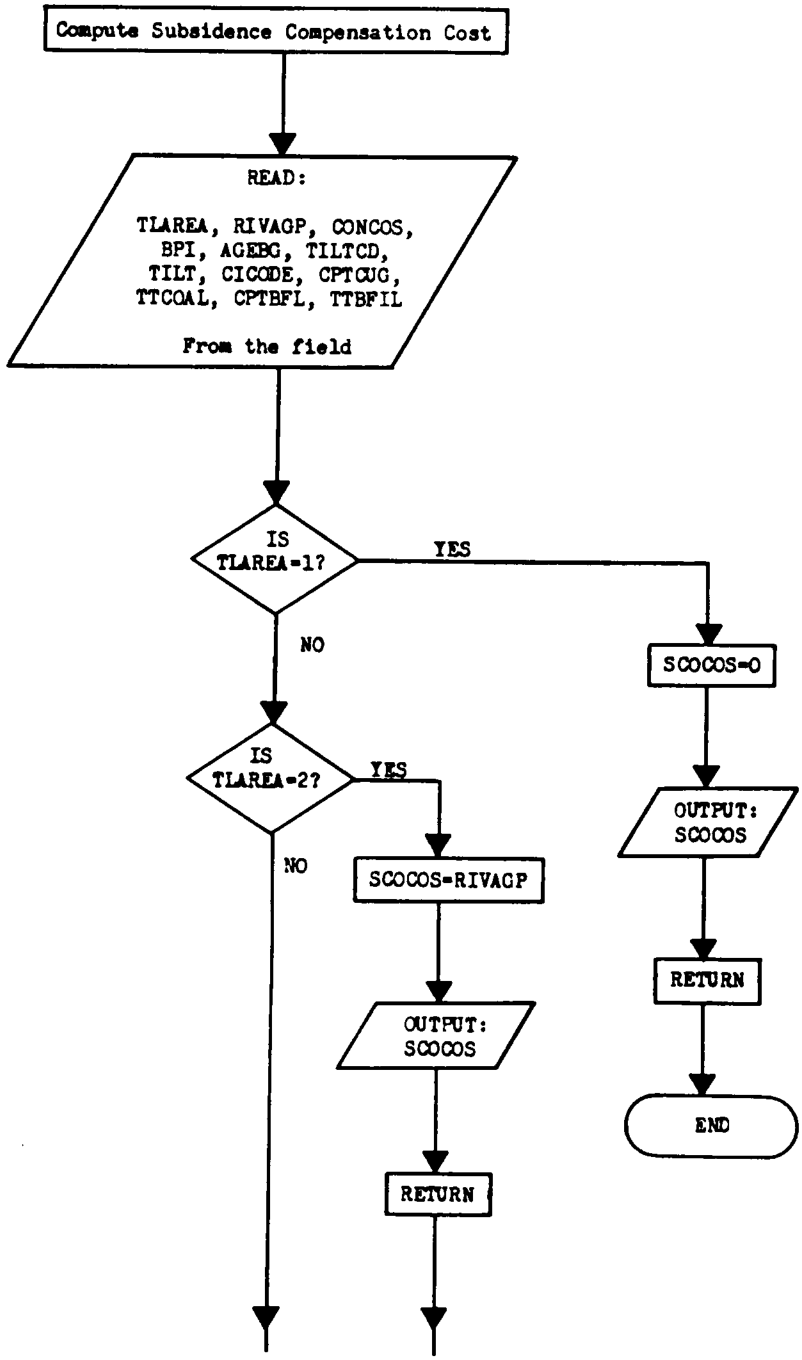

OUTPUT:
INDIRECT COST:
IDCOST
TOTAL: TTCOST
FIXED COST: FITC, DPCOST
GRAND TOTAL: GTCOST
DEPLETION ALLOWANCE:
DEPALL, EOCOST
FINAL TOTAL: FTCOST
INALLW=FTCOST*(((1+.055)**(UPYEAR-1977))-1)
OVCOST = FTCOST + INALLW
COSPAC = OVCOST/TTACRE
COSPT(NIND) = OVCOST/SEPROD
OUTPUT:
INFLATION ALLOWANCE: INALLW
Overall Cost of Secondary Mining:
OVCOST
Cost of Secondary Mining/Acre:
COSPAC
Cost of Pillar Extraction / Ton of Coal:
COSPT(NIND)

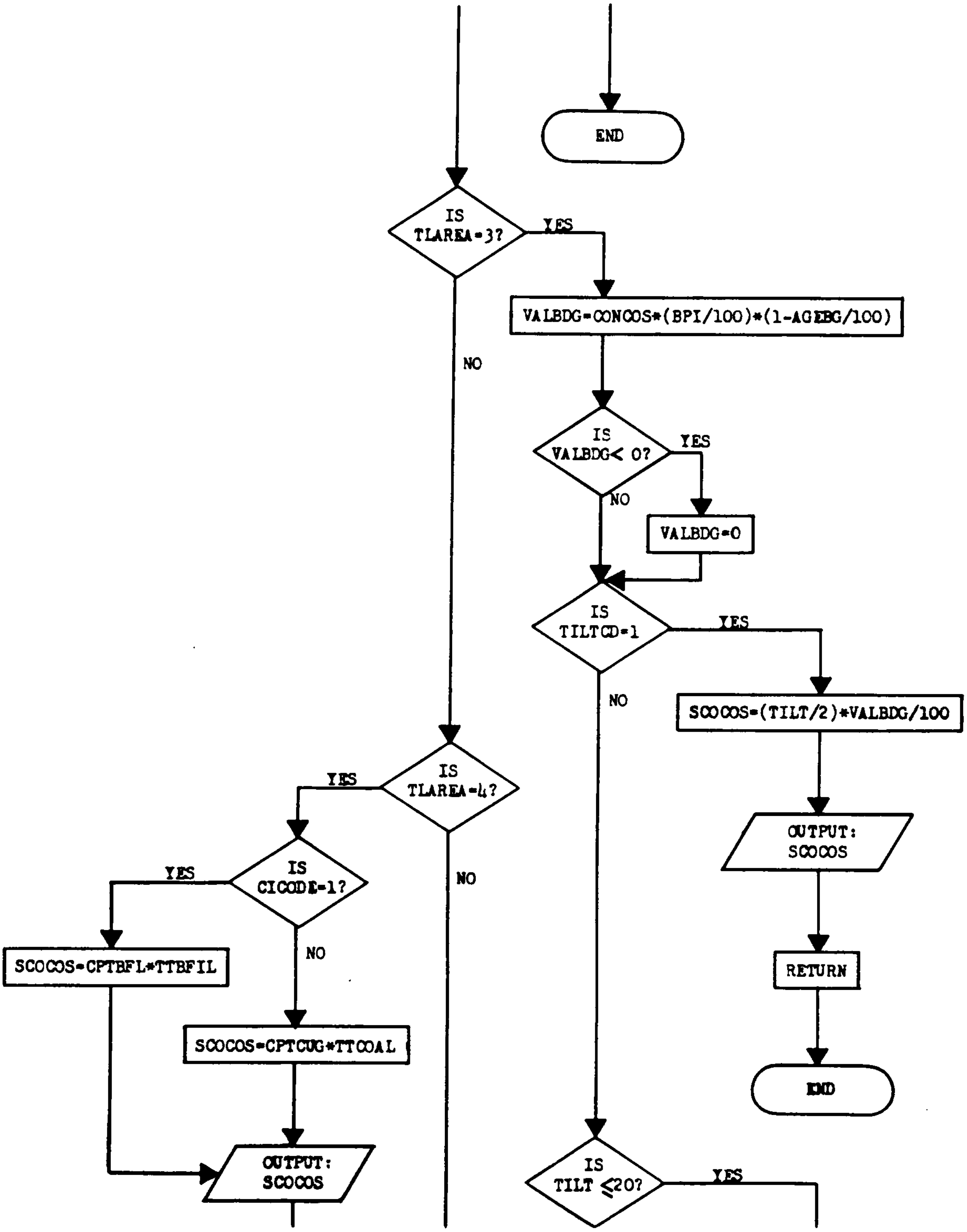


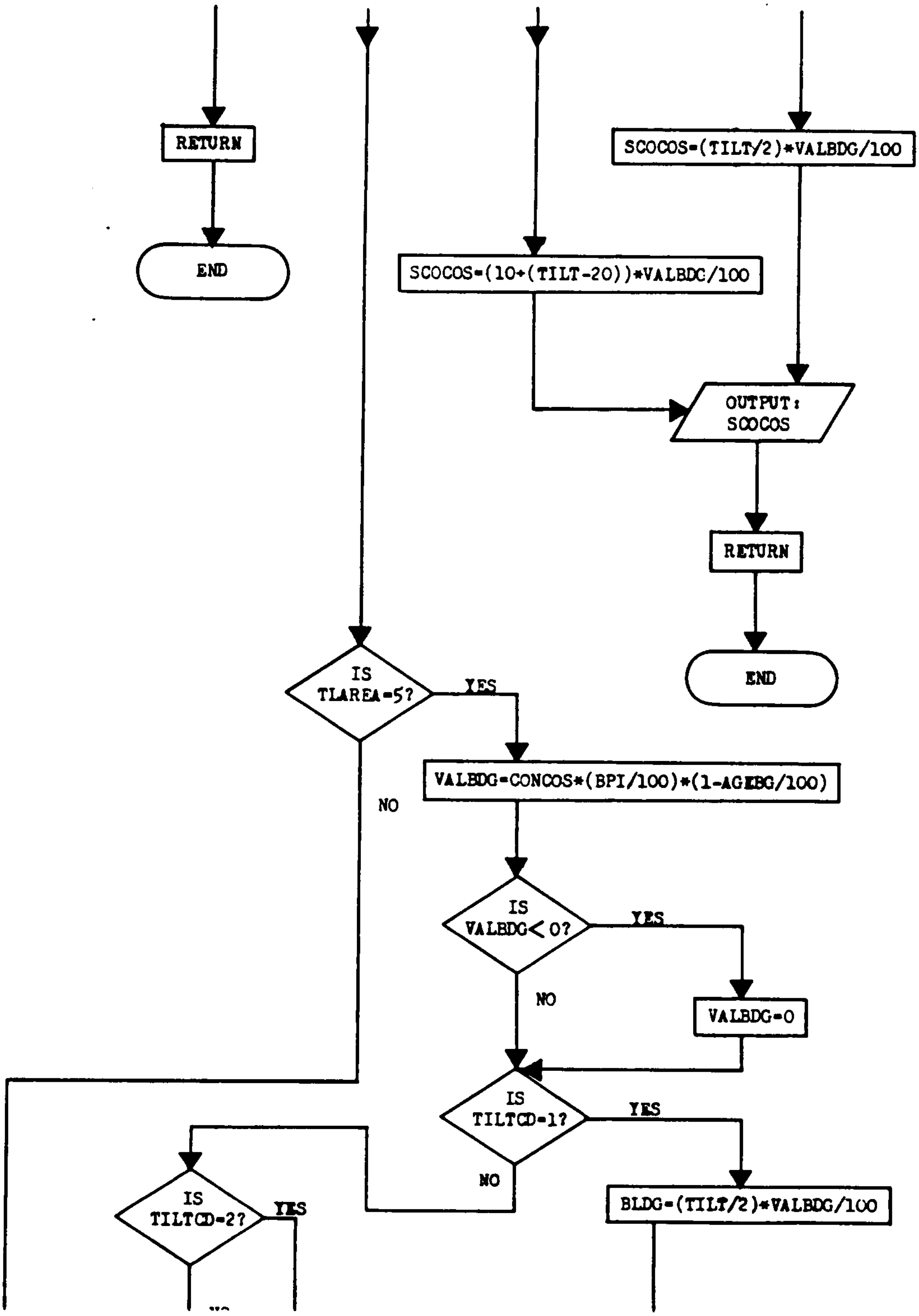
Detail Flowchart of Pillar Extraction Cost Simulator (PILCOST)

Figure A.2

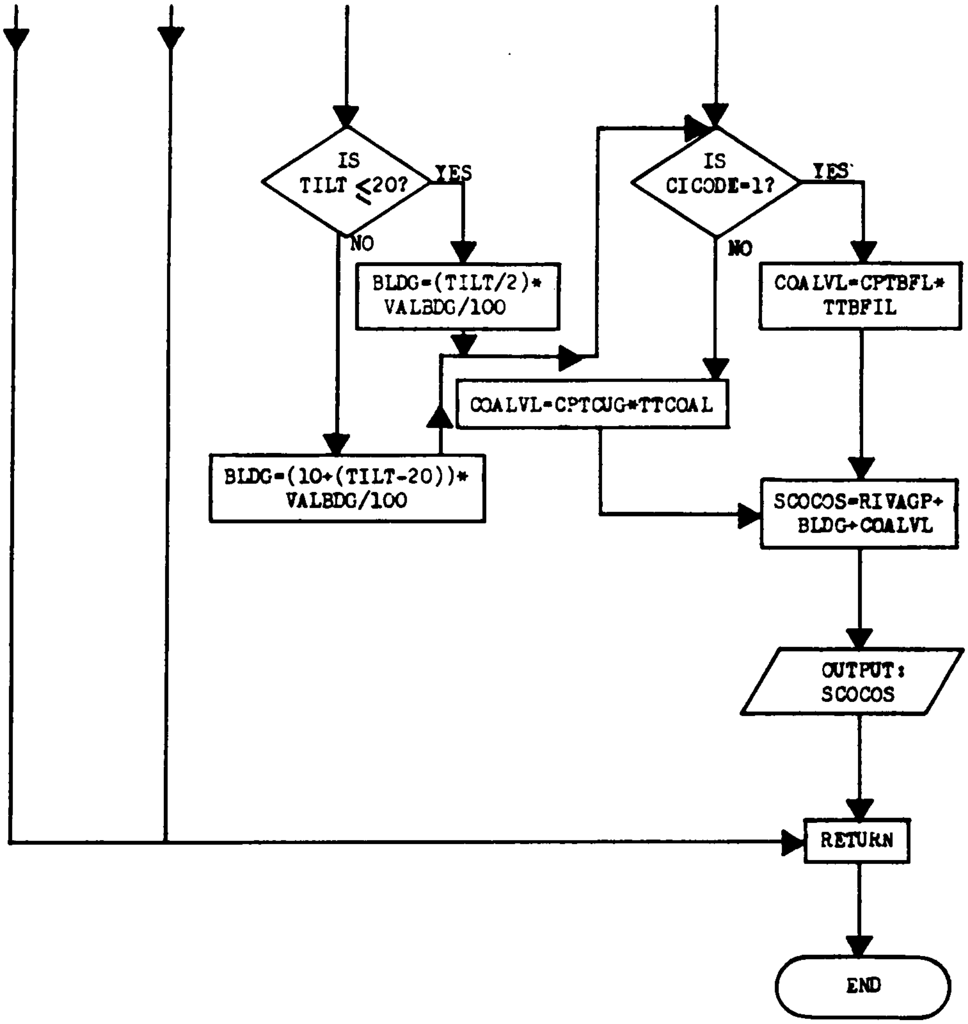
FLOWCHART FOR SUBSIDENCE COMPENSATION COST SUBROUTINE




END
IS
TLAREA=3?
YES
NO
VALBDG=CONCOS*(BPI/100)*(1-AGEBG/100)
IS
VALBDG< 0?
YES
NO
VALBDG=0
IS
TILTCD=1
YES
NO
SCOCOS=(TILT/2)*VALBDG/100
OUTPUT:
SCOCOS
RETURN
END
IS
TLAREA=4?
YES
NO
IS
CICODE=1?
YES
NO
SCOCOS=CPTBFL*TTBFIL
SCOCOS=CPTCUG*TTCOAL
OUTPUT:
SCOCOS
IS
TILT ≤20?
YES



RETURN
END
SCOCOS=(TILT/2)*VALBDG/100
SCOCOS=(10+(TILT-20))*VALBDC/100
OUTPUT:
SCOCOS
RETURN
END
IS
TLAREA=5?
YES
NO
VALBDG=CONCOS*(BPI/100)*(1-AGEBG/100)
IS
VALBDG< 0?
YES
NO
VALBDG=0
IS
TILTCD=1?
YES
NO
BLDG=(TILT/2)*VALBDG/100
IS
TILTCD=2?
YES



Detail Flowchart of SUBROUTINE SUBCOM.FOR.

Figure A.3

APPENDIX B

LISTINGS OF THE COMPUTER PROGRAM

LISTING OF COMPRESSIVE STRENGTH GENERATOR (COMSTREN.FOR)

```
C
C   THIS PROGRAM GENERATES COMPRESSIVE STRENGTH DATA USING THE
C   SAMPLE DATA COLLECTED FROM THE EXPERIMENTS IN THE LAB. THE
C   USER HAS TO SUPPLY THE PROBABILITY DISTRIBUTION CODE AFTER
C   TESTING THE TYPE OF PROBABILITY DISTRIBUTION. THE OUTPUT OF
C   THE PROGRAM GENERATES THE COMPRESSIVE STRENGTH DATA (30
C   DIFFERENT VALUES). THIS OUTPUT DATA IS USED AS INPUT TO MAIN
C   PROGRAM (PILCOST) TO COMPUTE PILLAR EXTRACTION COST VALUE.
C
      REAL KRAND,KRAND1,LAMDA,SIGCOM,LPROB(100),PI,LCOM(100)
      INTEGER SEED,STRENG,K,NA,NB,MINCOM,MAXCOM,MUCOM,CODE
      OPEN (4,FILE='COMDATA')
C
C   READ COMPRESSIVE STRENGTH AND CUMULATIVE PROBABILITY DATA
C   FOR EMPIRICAL DISTRIBUTION.
C
      N=0
    1 N=N+1
      READ(4,2)LCOM(N),LPROB(N)
      IF(LCOM(N))1,3,1
    3 NA=N-1
    2 FORMAT(F10.1,F10.3)
C
C   READ THE CODE TO SELECT PROBABILITY DISTRIBUTION:-
C      1 - EXPONENTIAL DISTRIBUTION
C      2 - UNIFORM DISTRIBUTION
C      3 - NORMAL DISTRIBUTION
C      4 - EMPIRICAL DISTRIBUTION
C   SEED - ANY FOUR DIGIT NUMBER TO GENERATE RANDOM NUMBERS.
C   IF UNIFORM DISTRIBUTION IS SELECTED ==>MINCOM - MINIMUM VALUE.
C   MAXCOM - MAXIMUM VALUE OF COMPRESSIVE STRENGTH.
C   IF NORMAL DISTRIBUTION IS SELECTED ==> MUCOM - AVERAGE VALUE.
C   SIGCOM - STANDARD DEVIATION OF COMPRESSIVE STRENGTH.
C   IF EXPONENTIAL DISTRIBUTION IS SELECTED ==>LAMDA-PARAMETER.
C   STRENG - COMPRESSIVE STRENGTH OF COAL SAMPLE.
C
C
      READ(4,4)CODE,SEED,MINCOM,MAXCOM,MUCOM
    4 FORMAT(5I5)
      READ(4,6)LAMDA,SIGCOM
    6 FORMAT(2F7.3)
      CLOSE(4)
      PI=3.1416
      NB=1
    5 K=7*SEED/16384
      SEED=7*SEED-16384*K
      KRAND=SEED/16384.0
      GO TO (10,11,12,13),CODE
```

```
C
C   GENERATOR FOR EXPONENTIAL DISTRIBUTION.
C
   10 STRENG=-ALOG(KRAND)/LAMDA
      GO TO 20
C
C   GENERATOR FOR UNIFORM DISTRIBUTION.
C
   11 STRENG=MINCOM+(MAXCOM-MINCOM)*KRAND
      GO TO 20
C
C   GENERATOR FOR NORMAL DISTRIBUTION.
C
   12 K=7*SEED/16384
      SEED=7*SEED-16384*K
      KRAND1=SEED/16384.0
      STRENG=MUCOM+SIGCOM*(SQRT(-2*ALOG(KRAND)))*COS(2*PI*KRAND1)
      GO TO 20
C
C   GENERATOR FOR EMPIRICAL DISTRIBUTION.
C
   13 DO 30 N=1,NA
      IF(LPROB(N)-KRAND)30,14,14
   14 STRENG=LCOM(N)
      GO TO 20
   30 CONTINUE
   20 NB=NB+1
      WRITE(*,21)STRENG
   21 FORMAT(5X,'THE COMPRESSIVE STRENGTH IS =',I6)
      IF(NB.LT.31)GO TO 5
      STOP
      END
```

LISTING OF THE PILLAR EXTRACTION COST SIMULATOR (PILCOST)

```
C         THIS PROGRAM IS DESIGNED TO COMPUTE ROOM AND PILLAR MINING
C       SECONDARY EXTRACTION COST. THE TOTAL ESTIMATED PRODUCTION COST
C       FOR PILLAR EXTRACTION CAN BE COMPUTED.
C         THE FIRST SECTION OF THE PROGRAM COMPUTES THE SAFE PILLAR
C       DIMENSION FOR GIVEN GEOLOGICAL AND MINING CONDITION. THE SEVERAL
C       VARIABLES USED IN PILLAR DESIGN EQUATION ARE AS FOLLOWS:
C         -----DEPTH OF COAL SEAM BELOW SURFACE
C         -----COAL SEAM THICKNESS (OR WORKING HEIGHT IN CASE OF THICK -
C                                                        COAL SEAM )
C         -----ROOM WIDTH
C         -----COAL STRENGTH
C         -----SAFETY FACTOR
C         THE GENERAL FORM OF PILLAR DESIGN EQUATION IS USED AND THE
C       USER HAS TO INPUT THE VALUES OF COEFFICIENTS OF PILLAR STRENGTH
C       FORMULA. THE GENERAL EQUATION OF PILLAR STRENGTH FORMULA CAN
C       COVER THE FOLLOWING FORMULAS OF PILLAR STRENGTH:
C              1). HOLLAND-GADDY FORMULA
C              2). MORRISON-CORLETT-RICE FORMULA
C              3). OBERT'S FORMULA
C              4). BIENIAWSKI'S FORMULA
C              5). SALAMON'S FORMULA
C       GENERAL FORM OF THE EQUATION IS PRESENTED BY :
C                                 Alpha  Beta
C              STRENGTH = C + K.W      .h
C
C       1. HOLLAND-GADDY FORMULA:
C              STRENGTH = K.SQRT(L)/T   in psi,
C                     K = Sp.SQRT(D)
C              Sp = STRENGTH OF CUBICAL SPECIMEN OF COAL OF DIMENSION D
C              L  = WIDTH OF PILLAR in inches,
C              T  = THICKNESS OF PILLAR  in inches,
C         THE VALUE OF DIFFERENT COEFFICIENTS OF GENERAL EQUATION FOR
C       HOLLAND-GADDY FORMULA WILL BE:
C                  C = 0
C              Alpha = 0.5
C               Beta = -1
C                  K = K.
C       2. MORRISON-CORLETT-RICE FORMULA:
C              STRENGTH = K.SQRT(W/T)  in psi.
C              W = Least Width of pillar,
C              T = Thickness of pillar,
C              K = Crushing Strength of 1" cube coal sample,
C         THE VALUE OF DIFFERENT COEFFICIENTS OF GENERAL EQUATION FOR
C       MORRISON-CORLETT-RICE FORMULA WILL BE:
C              C = 0
C          Alpha = 0.5
C           Beta = -0.5
C              K = K
```

```
C
C     3. OBERT'S FORMULA:
C             STRENGTH = C (0.778 + 0.222Wp/Hp)   in psi.
C                        1
C             C  = Average Strength of cubical coal specimens,
C              1
C             Wp = Least Width of pillar,
C             Hp = Height of pillar,
C          THE VALUE OF DIFFERENT COEFFICIENTS OF GENERAL EQUATION FOR
C     OBERT'S FORMULA WILL BE:
C               C = 0.778C
C                         1
C           Alpha = 1
C            Beta = -1
C               K = 0.222C
C                         1
C
C     4. BIENIAWSKI'S FORMULA:
C               STRENGTH = Sigma (0.645 + 0.355W/h)   in psi.
C                              C
C        Sigma  = Compressive Strength in psi for 5 feet cube coal,
C             C
C              W = Width of pillar in feet,
C              h = Working Height of pillar in feet,
C          THE VALUE OF DIFFERENT COEFFICIENTS OF GENERAL EQUATION FOR
C     BIENIAWSKI'S FORMULA WILL BE:
C               C = 0.645 Sigma
C                              C
C           Alpha = 1
C            Beta = -1
C               K = 0.355 Sigma
C                              C
C     5. SKELLY'S FORMULA: THIS IS SIMILAR TO BIENIAWSKI'S FORMULA.
C          THE VALUE OF DIFFERENT COEFFICIENTS OF GENERAL EQUATION FOR
C     SKELLY'S FORMULA WILL BE:
C               C = 0.78 Sigma
C                             C
C           Alpha = 1
C            Beta = -1
C               K = 0.22 Sigma
C                             C
C     6. SALAMON'S FORMULA:
C                                   .46  .66
C               STRENGTH = 1320 W   / h
C               W = Width of pillar,
C               h = Height of pillar,
C          THE VALUE OF DIFFERENT COEFFICIENTS OF GENERAL EQUATION FOR
C     SALAMON'S FORMULA WILL BE:
```

```
C             C = 0
C         Alpha = 0.46
C          Beta = -0.66
C             K =1320
C
C        THE USER HAS TO INPUT THE FOLLOWING VARIABLES TO COMPUTE THE
C     SAFE PILLAR DIMENSION:
C            ----DEPTH OF COAL SEAM BELOW SURFACE
C            ----SEAM THICKNESS (OR WORKING HEIGHT IN CASE OF THICK
C                                                         -COAL SEAM)
C            ----ROOM WIDTH
C            ----SAFETY FACTOR (DESIRED SAFETY FACTOR)
C            ----C
C            ----Alpha
C            ----Beta
C            ----K
C      THE OTHER VARIABLES USED IN THE PROGRAM STAND FOR THE FOLLOWING:
C          STRENG = Strength of coal pillar,
C          LOAD   = Load on coal pillar,
C          SAFETY = Computed safety factor,
C          SAFE   = Desired safety factor,
C          WIDTH  = Width of coal pillar (edge to edge),
C          HEIGHT = Height of coal pillar,
C          DEPTH  = Depth of coal seam below surface,
C          BORD   = Room width in the panel,
C          C      = Coefficient or constant in pillar strength equn.
C          K      =      "      "    "     "    "     "      "
C          ALPHA  = Power index for width of pillar,
C          BETA   = Power index for height of the pillar,
C          PREXTR = Primary mining extraction ratio,
C          PREX   = Primary mining percentage extraction,
C          SEXTRA = Secondary mining extraction ratio,
C          TIMBER = Total no. of timber props or headboards required
C                   to extract one pillar in retreat mining panel,
C          BOLT   = Total no. of roofbolts required to extract one
C                   pillar in retreat mining panel,
C          SECPER = Secondary mining percentage extraction (Maximum)
C          PRACRE = Tons mined per acre from primary mining,
C          MNLIFE = Primary mining life time in years,
C          YRPROD = Yearly primary mining production in tons,
C          TTACRE = Total acreage required for primary mining and
C                   then followed by secondary mining,
C          SEPROD = Extra coal mined during secondary mining in tons
C          PILTON = Coal mined from one pillar during secondary min.
C          NUMPIL = # Of Coal Pillars to be mined during secondary
C                   mining of coal,
C          RATIO  = Ratio of secondary to primary mining productivi-
C                   ties,
```

```
C          RATO   = Ratio of actual yearly production to base model
C                   production,
C          EXLIFE = Extra life of the mine using secondary mining
C                   method in years,
C          MONTHS = Extra life of mine in months,
C          TTLIFE = Total life of the mine using secondary mining
C                   (including primary mining life) in years,
C          DLCOST = Direct labor cost in dollars,
C          RTCOST = Rockbolt and timber cost in dollars,
C          MPCOST = Machine parts cost in dollars,
C          LHCOST = Lubrication and hydraulic oil cost in dollars,
C          RDCOST = Rock dust cost in dollars,
C          VSCOST = Ventilation supplies cost in dollars,
C          BTCOST = Bits cost in dollars,
C          CBCOST = Cables cost in dollars,
C          MSCOST = Miscellaneous cost in dollars,
C          PWCOST = Power Cost in dollars,
C          WTCOST = Water cost in dollars,
C          RLCOST = Royalty cost in dollars,
C          POCOST = Payroll overhead cost in dollars,
C          UWCOST = Union welfare cost in dollars,
C          RFCOST = Reclamation fund cost in dollars,
C          LICOST = Licences cost in dollars,
C          DPCOST = Depreciation cost in dollars,
C          IDCOST = Indirect cost in dollars,
C          TTCOST = Total cost (except depreciation cost) in dollars,
C          FITC   = Fixed Insurance & Taxes Cost in dollars,
C          GTCOST = Grand total cost in dollars,
C          DEPALL = Depletion allowance in dollars,
C          EOCOST = Extra operating cost due to thinning of seam by
C                   12 inches (from base of 72" to 60" USBM report)
C          ECINVT = Extra capital investment due to thinning of seam
C                   by 12" from base case (USBM report)
C          FTCOST = Final total cost (sum of grand total, depletion
C                   allowance, extra ope. cost, extra cap. invest.)
C          INALLW = Inflation Allowance (updating the total cost to
C                   current year from base year mid-1977 at 5.5% rate,
C          UPYEAR = Year for updating the total production cost from
C                   base year mid-1977,
C          OVCOST = Overall cost of secondary mining,
C          COSPAC = Cost of secondary mining per acre,
C          COSPT  = Cost of pillar extraction per ton of coal.
C
C
      DIMENSION COSPT(50)
      REAL STRENG,LOAD,HEIGHT,BORD,C,ALPHA,BETA,PREXTR,PREX,SEXTRA,
     1SECPER,PRACRE,TTACRE,PILTON,RATIO,RATO,EXLIFE,MONTHS,TTLIFE,
     2SAFETY,SAFE,K(50),SAFE1
```

```
      INTEGER WIDTH,DEPTH,TIMBER,BOLT,MNLIFE,YRPROD,SEPROD,NUMPIL,
     1DLCOST,RTCOST,MPCOST,LHCOST,RDCOST,VSCOST,BTCOST,CBCOST,MSCOST,
     2PWCOST,WTCOST,RLCOST,POCOST,UWCOST,RFCOST,LICOST,DPČOST,IDCOST,
     3TTCOST,FITC,GTCOST,EOCOST,FTCOST,INALLW,UPYEAR,TZCODE,
     4RESERV,OVCOST,COSPAC,SCOCOS,NIND
C
      COMMON /ONEP/ SCOCOS
      COMMON /TWOP/ EXLIFE,TIMBER,NUMPIL,MONTHS,BOLT,
     1GTCOST,EOCOST
C
      OPEN (4,FILE='PILDATA')
      READ(4,600)DEPTH,YRPROD,MNLIFE,UPYEAR,TZCODE
  600 FORMAT(5I7)
      READ(4,601)HEIGHT,BORD,SAFE,C,ALPHA,BETA,RATIO
  601 FORMAT(7F10.2)
      I=0
  602 I=I+1
      READ(4,604)K(I)
      IF(K(I))602,603,602
  603 N=I-1
  604 FORMAT(F7.1)
      COSPT(1)=99.0
      NIND=2
      SAFE1=SAFE
      DO 200 I=1,N
      SAFE=SAFE1
      WRITE(*,606)
  606 FORMAT(3X,'NEW OUTPUT CORRESPONDING TO NEXT STRENGTH DATA')
  133 DO 10 WIDTH=1,100
      STRENG=C+K(I)*(WIDTH**ALPHA)*(HEIGHT**BETA)
      LOAD=1.1*DEPTH*((1+BORD/WIDTH)**2)
      SAFETY=STRENG/LOAD
      IF(SAFETY.GE.SAFE) GO TO 11
   10 CONTINUE
   11 WRITE(*,12) WIDTH
   12 FORMAT(3X,'SAFE PILLAR DIMENSION IS ',I3,' FEET')
C
C
C        THE SAFE PILLAR WIDTH IS USED TO COMPUTE PRIMARY MINING
C     PERCENTAGE EXTRACTION.
C
C
      PREXTR=1-((WIDTH/(WIDTH+BORD))**2)
      PREX=100*PREXTR
      WRITE(*,9)PREX
    9 FORMAT(3X,'PRIMARY MINING PERCENTAGE EXTRACTION = ',F5.2)
C
C
```

```
C        THIS FOREGOING SECTION OF THE PROGRAM SELECTS THE PARTICULAR
C     PILLAR EXTRACTION METHOD BASED ON THE SAFE PILLAR DIMENSION.
C     THE RANGE OF SAFE PILLAR DIMENSION IS SHOWN HERE FOR THE
C     SELECTION OF PILLAR EXTRACTION METHOD.
C        PILLAR WIDTH RANGE            PILLAR EXTRACTION METHOD
C          1' - 13'                      Outside Lift (Single cut)
C         14' - 22'                      Outside Lift (Double cuts)
C         23' - 31'                      Modified Split & Fender
C         32' - <34'                     Christmas treeing method
C         34' - 55'                      Split and Fender
C        >55'                            Pocket and Fender
C
C
      IF(WIDTH.LE.13) GO TO 13
      IF(WIDTH.LE.22) GO TO 14
      IF(WIDTH.LE.31) GO TO 15
      IF(WIDTH.LT.34) GO TO 16
      IF(WIDTH.LE.43) GO TO 17
      IF(WIDTH.LE.55) GO TO 18
      IF(WIDTH.GT.55) GO TO 19
   13 WRITE(*,8)
    8 FORMAT(3X,'THE PILLAR EXTRACTION METHOD USED FOR THIS PILLAR',
     1' SIZE WILL BE "OUTSIDE LIFT WITH SINGLE CUT".')
      IF(WIDTH.LE.8) SEXTRA=.97
      IF((WIDTH.LE.10).AND.(WIDTH.GT.8)) SEXTRA=.96
      IF(WIDTH.EQ.11) SEXTRA=.95
      IF(WIDTH.GT.11) SEXTRA=.94
      TIMBER=28
      BOLT=0
C
C
C     TIMBER INCLUDES THE TOTAL NUMBER OF BREAKER POSTS AND TURN OR
C     FINGER PROPS REQUIRED TO EXTRACT ONE PILLAR IN SECONDARY MINING
C     SECTION. THE SAME NUMBER OF HEAD BOARDS ARE ALSO REQUIRED TO
C     EXTRACT ONE PILLAR WHICH ARE USED ABOVE THE TIMBER PROPS.
C
C
      GO TO 30
   14 WRITE(*,7)
    7 FORMAT(3X,'THE PILLAR EXTRACTION METHOD USED FOR THIS PILLAR',
     1' SIZE WILL BE "OUTSIDE LIFT WITH DOUBLE CUTS".')
      IF(WIDTH.LE.18) SEXTRA=.93
      IF(WIDTH.GT.18) SEXTRA=.92
      TIMBER=38
      BOLT=0
      GO TO 30
   15 WRITE(*,6)
    6 FORMAT(3X,'THE PILLAR EXTRACTION METHOD USED FOR THIS PILLAR',
```

```
  1' SIZE WILL BE "MODIFIED SPLIT AND FENDER METHOD".')
   IF(WIDTH.LE.24) GO TO 21
   IF(WIDTH.LE.27) GO TO 22
   IF(WIDTH.LE.30) GO TO 23
   IF(WIDTH.EQ.31) GO TO 24
21 SEXTRA=.91
   TIMBER=48
   BOLT=8
   GO TO 30
22 SEXTRA=.90
   TIMBER=49
   BOLT=10
   GO TO 30
23 SEXTRA=.895
   TIMBER=50
   BOLT=12
   GO TO 30
24 SEXTRA=.89
   TIMBER=51
   BOLT=14
   GO TO 30
16 WRITE(*,5)
 5 FORMAT(3X,'THE PILLAR EXTRACTION METHOD USED FOR THIS PILLAR',
  1' SIZE WILL BE "OUTSIDE-LIFT  CHRISTMAS TREEING".')
   SEXTRA=.86
   TIMBER=46
   BOLT=0
   GO TO 30
17 WRITE(*,4)
 4 FORMAT(3X,'THE PILLAR EXTRACTION METHOD USED FOR THIS PILLAR',
  1' SIZE WILL BE "SPLIT AND     FENDER METHOD".')
   SEXTRA=.90
   IF(WIDTH.EQ.34)GO TO 25
   IF(WIDTH.LE.38)GO TO 26
   IF(WIDTH.LE.43)GO TO 27
25 TIMBER=112
   BOLT=40
   GO TO 30
26 TIMBER=112
   BOLT=45
   GO TO 30
27 TIMBER=112
   BOLT=50
   GO TO 30
18 WRITE(*,3)
 3 FORMAT(3X,'THE PILLAR EXTRACTION METHOD USED FOR THIS PILLAR',
  1'SIZE WILL BE "SPLIT AND     FENDER METHOD".')
   SEXTRA=.85
```

```
      IF(WIDTH.LE.46)GO TO 28
      IF(WIDTH.LE.50)GO TO 29
      IF(WIDTH.LE.54)GO TO 31
      IF(WIDTH.EQ.55)GO TO 32
   28 TIMBER=126
      BOLT=50
      GO TO 30
   29 TIMBER=126
      BOLT=55
      GO TO 30
   31 TIMBER=126
      BOLT=60
      GO TO 30
   32 TIMBER=126
      BOLT=65
      GO TO 30
   19 WRITE(*,2)
    2 FORMAT(3X,'THE PILLAR EXTRACTION METHOD USED FOR THIS PILLAR',
     1' SIZE WILL BE "POCKET AND WING METHOD".')
      IF(WIDTH.LE.60)GO TO 52
      IF(WIDTH.LE.65)GO TO 53
      IF(WIDTH.LE.73)GO TO 54
      IF(WIDTH.LE.78)GO TO 55
      IF(WIDTH.LE.85)GO TO 56
      IF(WIDTH.GT.85)GO TO 130
   52 SEXTRA=.84
      TIMBER=186
      BOLT=135
      GO TO 30
   53 SEXTRA=.82
      TIMBER=200
      BOLT=155
      GO TO 30
   54 SEXTRA=.81
      TIMBER=217
      BOLT=190
      GO TO 30
   55 SEXTRA=.80
      TIMBER=233
      BOLT=205
      GO TO 30
   56 SEXTRA=.79
      TIMBER=314
      BOLT=230
      GO TO 30
C
C
C
```

```
C          THE COAL RESERVE IS BASED ON 1800 TONS OF COAL PER-ACRE-FOOT
C      AND A BASE MODEL CASE OF 5 FEET THICK SEAM IS CONSIDERED FOR
C      ANALYSIS. SURFACE AREA REQUIRED TO PROVIDE COAL RESOURCES NECE-
C      SSARY TO SUSTAIN A GIVEN PERIOD OF PRIMARY MINING OPERATION CAN
C      BE COMPUTED.
C          THE USER HAS TO INPUT THE DATA RELATED TO PRODUCTIVITIES AND
C      THE INPUT VARIABLES ARE LISTED AS FOLLOWS:
C            YRPROD = Annual primary mining production in tons,
C            MNLIFE = Primary mining life in years,
C            RATIO  = Ratio of secondary to primary mining productivity
C            UPYEAR = Year for updating the total production cost from
C                     base year mid-1977,
C          AMOUNT OF COAL PER-ACRE FOR A 5' THICK COAL SEAM = 9000 TONS
C      THEREFORE, MINED COAL PER ACRE (USING DEVELOPMENT ONLY) CAN BE
C      COMPUTED AS FOLLOWS:
C
C
   30 SECPER=SEXTRA*100
      WRITE(*,1)SECPER
    1 FORMAT(3X,'SECONDARY MINING PERCENTAGE EXTRACTION (MAX.) = ',F5.2)
      WRITE(*,91)TIMBER
   91 FORMAT(3X,'TOTAL TIMBER POSTS REQUIRED TO EXTRACT ONE PILLAR = ',
     1I3)
      WRITE(*,92)TIMBER
   92 FORMAT(3X,'TOTAL TIMBER HEADBOARDS REQUIRED TO EXTRACT ONE',
     1' PILLAR = ',I3)
      WRITE(*,93)BOLT
   93 FORMAT(3X,'TOTAL NUMBER OF ROCKBOLTS REQUIRED TO EXTRACT ONE',
     1' PILLAR = ',I3)
      RESERV=1800*HEIGHT
      WRITE(*,94)RESERV
   94 FORMAT(3X,'TOTAL AMOUNT OF COAL RESERVE PER ACRE (VIRGIN',
     1' PROPERTY) IN TONS = ',I5)
      PRACRE=1800*PREXTR*HEIGHT
      WRITE(*,95)PRACRE
   95 FORMAT(3X,'TONS MINED PER ACRE FROM PRIMARY MINING = ',F8.2)
      TTACRE=YRPROD*MNLIFE/PRACRE
      WRITE(*,96)TTACRE
   96 FORMAT(3X,'TOTAL ACREAGE REQUIRED FOR PRIMARY MINING = ',F8.2)
      SEPROD=1800*TTACRE*(SEXTRA-PREXTR)*HEIGHT
      WRITE(*,97)SEPROD
   97 FORMAT(3X,'EXTRA COAL MINED DURING SECONDARY MINING = ',I10)
      PILTON=(SEXTRA-PREXTR)*(WIDTH*WIDTH*HEIGHT*86/2000)/(1-PREXTR)
      WRITE(*,98)PILTON
   98 FORMAT(3X,'TOTAL COAL MINED BY EXTRACTING ONE PILLAR = ',F6.1)
      NUMPIL=(SEPROD/PILTON)+.5
      WRITE(*,99)NUMPIL
   99 FORMAT(3X,'TOTAL NUMBER OF PILLARS TO BE EXTRACTED = ',I8)
```

```
C
C
C       THE PRODUCTIVITY BECOMES MORE DURING SECONDARY MINING AND
C     USER HAS TO INPUT THE VALUE OF RATIO OF SECONDARY TO PRIMARY
C     MINING PRODUCTIVITIES. THIS VALUE VARIES FROM MINE TO MINE
C     DEPENDING ON SEVERAL FACTORS LIKE MAINTENANCE OF ENTRIES ETC.,
C
C
      EXLIFE=SEPROD/(RATIO*YRPROD)
      WRITE(*,100)EXLIFE
  100 FORMAT(3X,'EXTENDED LIFE OF MINE IN YEARS = ',F5.3)
      MONTHS=12*EXLIFE
      WRITE(*,101)MONTHS
  101 FORMAT(3X,'EXTENDED LIFE OF MINE IN MONTHS = ',F6.2)
      TTLIFE=MNLIFE+EXLIFE
      WRITE(*,102)TTLIFE
  102 FORMAT(3X,'TOTAL LIFE OF MINE IN YEARS (INCLUDING SECONDARY',
     1' MINING) = ',F6.3)
C
C       THIS SECTION OF THE MAIN PROGRAM COMPUTES THE VARIOUS OPERA-
C     TING COSTS. ESTIMATED YEARLY OPERAING, DEPRECIATION, DEPLETION
C     ALLOWANCE, SUBSIDENCE COMPENSATION COSTS ETC. ARE COMPUTED FOR
C     A BASE CASE OF 5 FEET THICK COAL SEAM. THE COSTS ARE ADJUSTED
C     AND UPDATED FOR ANY SEAM THICKNESS USING THE ADJUSTMENT FACTORS
C     AND BASE CASE OF MID-1977.
C       THE YEARLY COST DATA ARE DIFFERENT FOR DIFFERENT YEARLY
C     PRODUCTION RATES. SEVERAL YEARLY PRODUCTION RATES AND CORRES-
C     PONDING COSTS HAVE BEEN CONSIDERED AND THE CLOSEST COST DATA
C     CAN BE USED.
C
C
C               '
C               '
C               '
C               '
C               '
      IF(TZCODE.EQ.1)GO TO 70
      IF(YRPROD.GE.750000)GO TO 41
      IF(YRPROD.GE.250000)GO TO 42
      GO TO 43
C
C
C       THIS SECTION OF THE MAIN PROGRAM COMPUTES THE OPERATING COST
C     FOR A COAL MINE PRODUCING 1 MILLION TON PER TON PER YEAR. IF THE
C     COAL PRODUCTION FOR THE MINE CONCERNED IS LESS THAN OR GREATER
C     THAN 1 MILLION TON, IT CAN BE  ADJUSTED TO ACTUAL VALUE USING
C     COST ADJUSTMENT FACTORS.
C
```

```
C
   41 RATO=YRPROD/1000000.0
      DLCOST=3921500*EXLIFE*RATO
      RTCOST=(5.42*TIMBER*NUMPIL)+(57.98*90*MONTHS)+(2*BOLT*NUMPIL)+
     1(10.04*90*MONTHS)
      MPCOST=770300*EXLIFE*RATIO*RATO
      LHCOST=305900*EXLIFE*RATIO*RATO
      RDCOST=165000*EXLIFE*RATIO*RATO
      VSCOST=237700*EXLIFE*RATIO*RATO
      BTCOST=150100*EXLIFE*RATIO*RATO
      CBCOST=73100*EXLIFE*RATIO*RATO
      MSCOST=182900*EXLIFE*RATIO*RATO
      PWCOST=506500*EXLIFE*RATIO*RATO
      WTCOST=3000*EXLIFE*RATIO*RATO
      RLCOST=200000*EXLIFE*RATIO*RATO
      POCOST=0.40*DLCOST
      UWCOST=1443400*EXLIFE*RATO
      RFCOST=150000*EXLIFE*RATO
      LICOST=100000*EXLIFE*RATO
      DPCOST=1256000*EXLIFE
      GO TO 50
C
C
C        THIS SECTION OF THE MAIN PROGRAM COMPUTES THE OPERATING COST
C     FOR A COAL MINE PRODUCING 500,000 TONS OF COAL PER YEAR. IF THE
C     YEARLY PRODUCTION OF COAL IN  THE  CONCERNED MINE IS BETWEEN
C     250,000 - 750,000 TONS, THIS SECTION OF THE PROGRAM IS EXECUTED
C     AND THE COSTS ARE ADJUSTED FOR ACTUAL PRODUCTION RATE.
C
C
   42 RATO=YRPROD/500000.0
      DLCOST=2377900*EXLIFE*RATO
      RTCOST=(5.42*TIMBER*NUMPIL)+(57.98*90*MONTHS)+(2*BOLT*NUMPIL)
     1+(10.04*90*MONTHS)
      MPCOST=385200*EXLIFE*RATIO*RATO
      LHCOST=153000*EXLIFE*RATIO*RATO
      RDCOST=82500*EXLIFE*RATIO*RATO
      VSCOST=118900*EXLIFE*RATIO*RATO
      BTCOST=75100*EXLIFE*RATIO*RATO
      CBCOST=36600*EXLIFE*RATIO*RATO
      MSCOST=91500*EXLIFE*RATIO*RATO
      PWCOST=281600*EXLIFE*RATIO*RATO
      WTCOST=1400*EXLIFE*RATIO*RATO
      RLCOST=100000*EXLIFE*RATIO*RATO
      POCOST=0.40*DLCOST
      UWCOST=759600*EXLIFE*RATO
      RFCOST=75000*EXLIFE*RATO
      LICOST=50000*EXLIFE*RATO
```

```
      DPCOST=838100*EXLIFE
      GO TO 50
C
C
C        THIS SECTION OF THE MAIN PROGRAM COMPUTES THE OPERATING COST
C     FOR A COAL MINE PRODUCING 150,000 TONS OF COAL PER YEAR. IF THE
C     YEARLY PRODUCTION OF COAL IN THE MINE CONCERNED IS LESS THAN
C     250,000 TONS, THIS SECTION OF THE PROGRAM IS EXECUTED AND THE
C     COSTS ARE ADJUSTED FOR ACTUAL PRODUCTION.
C
C
   43 RATO=YRPROD/150000.0
      DLCOST=945300*EXLIFE*RATO
      RTCOST=(5.42*TIMBER*NUMPIL)+(57.98*90*MONTHS)+(2*BOLT*NUMPIL)+
     1(10.04*90*MONTHS)
      MPCOST=115500*EXLIFE*RATIO*RATO
      LHCOST=45900*EXLIFE*RATIO*RATO
      RDCOST=24800*EXLIFE*RATIO*RATO
      VSCOST=35700*EXLIFE*RATIO*RATO
      BTCOST=22500*EXLIFE*RATIO*RATO
      CBCOST=11000*EXLIFE*RATIO*RATO
      MSCOST=27400*EXLIFE*RATIO*RATO
      PWCOST=119900*EXLIFE*RATIO*RATO
      WTCOST=400*EXLIFE*RATIO*RATO
      RLCOST=30000*EXLIFE*RATIO*RATO
      POCOST=0.40*DLCOST
      UWCOST=255800*EXLIFE*RATO
      RFCOST=22500*EXLIFE*RATO
      LICOST=15000*EXLIFE*RATO
      DPCOST=296260*EXLIFE
   50 WRITE(*,127)
  127 FORMAT(3X,'     ')
      WRITE(*,103)
  103 FORMAT(34X,'ESTIMATED TOTAL PRODUCTION COST')
      WRITE(*,127)
      WRITE(*,104)
  104 FORMAT(16X,'DIRECT COST:')
      WRITE(*,127)
      WRITE(*,105)
  105 FORMAT(21X,'Production & Maintenance:')
      WRITE(*,106)DLCOST
  106 FORMAT(27X,'Labor and Supervision                   $',I10)
      WRITE(*,107)
  107 FORMAT(21X,'Operating Supplies:')
      WRITE(*,108)RTCOST,MPCOST,LHCOST,RDCOST,VSCOST,BTCOST,CBCOST,
     1MSCOST
  108 FORMAT(27X,'Rockbolts and Timbers                   $',I10/
     +       27X,'Mining Machine Parts                    $',I9/
```

```
+        27X,'Lubrication & Hydraulic Oil            $',I9/
+        27X,'Rock Dust                              $',I9/
+        27X,'Ventilation                            $',I9/
+        27X,'Bits                                   $',I9/
+        27X,'Cables                                 $',I9/
+        27X,'Miscellaneous                          $',I9)
      WRITE(*,109)PWCOST,WTCOST,RLCOST,POCOST,UWCOST,RFCOST,LICOST
 109 FORMAT(21X,'Power                                       $',I9/
+        21X,'Water                                      $',I9/
+        21X,'Royalty                                    $',I9/
+        21X,'Payroll Overhead (40% of Payroll)          $',I10/
+        21X,'Union Welfare                              $',I9/
+        21X,'Reclamation Fund                           $',I9/
+        21X,'Licenses                                   $',I8)
      WRITE(*,127)
      WRITE(*,110)
 110 FORMAT(16X,'INDIRECT COST:')
      WRITE(*,127)
      IDCOST=0.15*(DLCOST+RTCOST+MPCOST+LHCOST+RDCOST+VSCOST+BTCOST+
     1CBCOST+MSCOST)
      WRITE(*,111)IDCOST
 111 FORMAT(21X,'15% of Labor,Supervision, & Supplies         $',I10)
      WRITE(*,112)
 112 FORMAT(16X,'_____________________________________________',
     1'__________')
      TTCOST=DLCOST+RTCOST+MPCOST+LHCOST+RDCOST+VSCOST+BTCOST+CBCOST+
     1MSCOST+PWCOST+WTCOST+RLCOST+POCOST+UWCOST+RFCOST+LICOST+IDCOST
      WRITE(*,113)TTCOST
 113 FORMAT(50X,'TOTAL          $',I12)
      WRITE(*,112)
      WRITE(*,127)
      WRITE(*,114)
 114 FORMAT(16X,'FIXED COST:')
      WRITE(*,127)
      FITC=0.02*TTCOST
      WRITE(*,115)FITC,DPCOST
 115 FORMAT(21X,'Taxes and Insurance (2% of Mine Cost)        $',I10/
+        21X,'Depreciation                               $',I10)
      WRITE(*,112)
      GTCOST=TTCOST+FITC+DPCOST
      WRITE(*,116)GTCOST
 116 FORMAT(21X,'GRAND TOTAL (Fixed, Direct,& Indirect)       $',I12)
      WRITE(*,112)
      WRITE(*,127)
      EOCOST=175000*EXLIFE*(6-HEIGHT)
      WRITE(*,118)
 118 FORMAT(16X,'EXTRA OPERATING COST DUE TO THINNING OF SEAM')
      WRITE(*,119)EOCOST
```

```
119 FORMAT(19X,'(Deviation from USBM model of 6 feet thick seam)  $',
   1I8)
    GO TO 71
 70 CALL USER
 71 CALL SUBSID
    WRITE(*,112)
    FTCOST=GTCOST+EOCOST+SCOCOS
    WRITE(*,122)FTCOST
122 FORMAT(21X,'FINAL TOTAL                                    $',I12)
    WRITE(*,112)
    INALLW=FTCOST*(((1+.055)**(UPYEAR-1977))-1)
    WRITE(*,127)
    WRITE(*,123)INALLW
123 FORMAT(16X,'INFLATION ALLOWANCE                                 $',
   1I10)
    OVCOST=FTCOST+INALLW
    WRITE(*,112)
    WRITE(*,124)OVCOST
124 FORMAT(21X,'OVERALL COST OF RETREAT MINING                 $',I12)
    WRITE(*,112)
    WRITE(*,127)
    COSPAC=OVCOST/TTACRE
    WRITE(*,125)COSPAC
125 FORMAT(16X,'COST OF RETREAT MINING (PILLAR EXTRACT.) PER ACRE= $',
   1I6)
    WRITE(*,127)
    COSPT(NIND)=OVCOST/SEPROD
    WRITE(*,126)COSPT(NIND)
    IF((COSPT(NIND).LT.COSPT(NIND-1)).AND.(DEPTH.LT.1500))GO TO 131
    GO TO 200
131 NIND=NIND+1
    SAFE=SAFE+0.1
    GO TO 133
126 FORMAT(16X,'COST OF PILLAR EXTRACTION PER TON OF COAL = $',F5.2)
200 CONTINUE
130 STOP
    END
```

LISTING OF SUBROUTINE COST.FOR PROGRAM

```
      SUBROUTINE USER
C
C     This subroutine computes the total pillar extraction cost based
C  on the unit cost parameters (cost per ton of pillar extraction)
C  supplied by the user.
C
C
C
C
C
      INTEGER YRPROD,TIMBER,NUMPIL,BOLT,DLCOST,RTCOST,MPCOST,LHCOST,
     1RDCOST,VSCOST,BTCOST,CBCOST,MSCOST,PWCOST,WTCOST,RLCOST,POCOST,
     2UWCOST,RFCOST,LICOST,DPCOST,IDCOST,TTCOST,FITC,GTCOST,EOCOST
C
C
      REAL EXLIFE,RATIO,MONTHS,DLCOS1,MPCOS1,LHCOS1,RDCOS1,VSCOS1,
     1BTCOS1,CBCOS1,MSCOS1,PWCOS1,WTCOS1,RLCOS1,UWCOS1,RFCOS1,LICOS1,
     2DPCOS1
C
C
      COMMON /TWOP/ EXLIFE,TIMBER,NUMPIL,MONTHS,BOLT,
     1GTCOST,EOCOST
C
C
      OPEN(4,FILE='USEDATA')
      READ(4,5)DLCOS1,MPCOS1,LHCOS1,RDCOS1,VSCOS1,BTCOS1,CBCOS1,MSCOS1,
     1PWCOS1,WTCOS1,RLCOS1,UWCOS1,RFCOS1,LICOS1,DPCOS1
 5    FORMAT(15F5.2)
      READ(4,6)YRPROD
 6    FORMAT(I7)
      READ(4,7)RATIO
 7    FORMAT(F5.1)
      CLOSE(4)
C
C
      DLCOST=DLCOS1*YRPROD*EXLIFE
      RTCOST=(5.42*TIMBER*NUMPIL)+(57.98*90*MONTHS)+(2*BOLT*NUMPIL)+
     1(10.04*90*MONTHS)
      MPCOST=MPCOS1*YRPROD*EXLIFE*RATIO
      LHCOST=LHCOS1*YRPROD*EXLIFE*RATIO
      RDCOST=RDCOS1*YRPROD*EXLIFE*RATIO
      VSCOST=VSCOS1*YRPROD*EXLIFE*RATIO
      BTCOST=BTCOS1*YRPROD*EXLIFE*RATIO
      CBCOST=CBCOS1*YRPROD*EXLIFE*RATIO
      MSCOST=MSCOS1*YRPROD*EXLIFE*RATIO
      PWCOST=PWCOS1*YRPROD*EXLIFE*RATIO
      WTCOST=WTCOS1*YRPROD*EXLIFE*RATIO
      RLCOST=RLCOS1*YRPROD*EXLIFE*RATIO
```

```
      POCOST=0.40*DLCOST
      UWCOST=UWCOS1*YRPROD*EXLIFE
      RFCOST=RFCOS1*YRPROD*EXLIFE
      LICOST=LICOS1*YRPROD*EXLIFE
      DPCOST=DPCOS1*YRPROD*EXLIFE
      WRITE(*,11)
11    FORMAT(3X,'     ')
      WRITE(*,12)
12    FORMAT(34X,'ESTIMATED TOTAL PRODUCTION COST')
      WRITE(*,11)
      WRITE(*,13)
13    FORMAT(16X,'DIRECT COST:')
      WRITE(*,11)
      WRITE(*,14)
14    FORMAT(21X,'Production & Maintenance:')
      WRITE(*,15)DLCOST
15    FORMAT(27X,'Labor and Supervision                   $',I10)
      WRITE(*,16)
16    FORMAT(21X,'Operating Supplies:')
      WRITE(*,17)RTCOST,MPCOST,LHCOST,RDCOST,VSCOST,BTCOST,CBCOST,
     1MSCOST
17    FORMAT(27X,'Rockbolts and Timbers                   $',I10/
     +       27X,'Mining Machine Parts                    $',I9/
     +       27X,'Lubrication & Hydraulic Oil             $',I9/
     +       27X,'Rock Dust                               $',I9/
     +       27X,'Ventilation                             $',I9/
     +       27X,'Bits                                    $',I9/
     +       27X,'Cables                                  $',I9/
     +       27X,'Miscellaneous                           $',I9)
      WRITE(*,18)PWCOST,WTCOST,RLCOST,POCOST,UWCOST,RFCOST,LICOST
18    FORMAT(21X,'Power                                            $',I9/
     +       21X,'Water                                            $',I9/
     +       21X,'Royalty                                          $',I9/
     +       21X,'Payroll Overhead (40% of Payroll)                $',I10/
     +       21X,'Union Welfare                                    $',I9/
     +       21X,'Reclamation Fund                                 $',I9/
     +       21X,'Licenses                                         $',I8)
      WRITE(*,11)
      WRITE(*,19)
19    FORMAT(16X,'INDIRECT COST:')
      WRITE(*,11)
      IDCOST=0.15*(DLCOST+RTCOST+MPCOST+LHCOST+RDCOST+VSCOST+BTCOST
     1+CBCOST+MSCOST)
      WRITE(*,20)IDCOST
20    FORMAT(21X,'15% OF Labor, Supervision, & Supplies        $',I10)
      WRITE(*,21)
21    FORMAT(16X,'_____________________________________________',
     1'__________')
```

```
      TTCOST=DLCOST+RTCOST+MPCOST+LHCOST+RDCOST+VSCOST+BTCOST+CBCOST
     1+MSCOST+PWCOST+WTCOST+RLCOST+POCOST+UWCOST+RFCOST+LICOST+IDCOST
      WRITE(*,22)TTCOST
22    FORMAT(50X,'TOTAL              $',I12)
      WRITE(*,21)
      WRITE(*,11)
      WRITE(*,23)
 23   FORMAT(16X,'FIXED COST:')
      WRITE(*,11)
      FITC=0.02*TTCOST
      WRITE(*,24)FITC,DPCOST
 24   FORMAT(21X,'Taxes & Insurance (2% of Mine Cost)         $',I10/
     +       21X,'Depreciation                                $',I10)
      WRITE(*,21)
      GTCOST=TTCOST+FITC+DPCOST
      WRITE(*,25)GTCOST
 25   FORMAT(21X,'GRAND TOTAL (Fixed, Direct, & Indirect)     $',I12)
      WRITE(*,21)
      WRITE(*,11)
      EOCOST=0
      RETURN
      END
```

LISTING OF SUBROUTINE SUBCOM.FOR PROGRAM

```
      SUBROUTINE SUBSID
C
C     This subroutine computes the subsidence compensation cost. There
C    is a list of subsidence damages like tilting of building, damage
C    agricultural land, damage to surface communication installations,
C    etc.,
C
      INTEGER TLAREA,SCOCOS,RIVAGP,CONCOS,BPI,AGEBG,VALBDG,TILTCD
      INTEGER TILT,CICODE,TTCOAL,TTBFIL,BLDG,COALVL
      REAL CPTCUG,CPTBFL
      COMMON /ONEP/ SCOCOS
C
C     TLAREA  ---- CODE FOR TYPE OF SURFACE LAND
C               1 -  BARREN AREA,
C               2 -  AGRICULTURAL LAND AREA,
C               3 -  BUILDINGS IN THE AREA,
C               4 -  COMMUNICATION INSTALLATIONS IN THE AREA,
C               5 -  COMBINED (AGRICULTURAL, BUILDINGS, COMMUNICATION
C                          INSTALLATIONS IN THE AREA).
C     SCOCOS - Subsidence Compensation Cost,
C     RIVAGP - Reduction in Value of Agricultural Property,
C     CONCOS - Construction Cost of Building in 1913,
C     BPI    - Building Price Index based on age of building,
C     AGEBG  - Age of Building,
C     VALBDG - Value of Building to be replaced on the target date of
C              mining subsidence,
C     TILTCD - Tilt Code to choose one of the methods of computing
C              loss in value of a building due to subsidence,
C              1 - Vennhofen's Method,
C              2 - Agreed Basis,
C     TILT   - Tilt Value in mm/Metre,
C     CICODE - Code to choose backfilling method or leaving coal
C              pillars underground to protect surface installations,
C     CPTCUG - Cost per ton of coal left underground,
C     TTCOAL - Total tons of coal left underground to protect surface
C              communication installations,
C     CPTBFL - Cost per ton of backfilling material required,
C     TTBFIL - Total backfilling material required to protect communi-
C              cation installations,
C     BLDG   - Subsidence Cost due to tilting of building,
C     COALVL - Dollar Value of Coal left underground to protect surface
C              communication installations.
      OPEN(4,FILE='SUBSIDAT')
      READ(4,401) TLAREA,RIVAGP,CONCOS,BPI,AGEBG,TILTCD,TILT
  401 FORMAT(7I10)
      READ(4,402) CICODE,CPTCUG,TTCOAL,CPTBFL,TTBFIL
  402 FORMAT(I1,F5.2,I10,F5.2,I10)
      CLOSE(4)
C
```

```
C
      GO TO (410,420,430,440,450),TLAREA
  410 SCOCOS=0
      GO TO 460
  420 SCOCOS=RIVAGP
      GO TO 460
  430 VALBDG=CONCOS*(BPI/100)*(1-AGEBG/100)
      IF(VALBDG.LT.0) VALBDG=0
      GO TO (431,432),TILTCD
  431 SCOCOS=(TILT/2)*VALBDG/100
      GO TO 460
  432 IF(TILT.LE.20) GO TO 433
      SCOCOS=(10+(TILT-20))*VALBDG/100
      GO TO 460
  433 SCOCOS=(TILT/2)*VALBDG/100
      GO TO 460
  440 IF (CICODE.EQ.1) GO TO 441
      SCOCOS=CPTCUG*TTCOAL
      GO TO 460
  441 SCOCOS=CPTBFL*TTBFIL
      GO TO 460
C
  450 VALBDG=CONCOS*(BPI/100)*(1-AGEBG/100)
      IF(VALBDG.LT.0) VALBDG=0
      GO TO (451,452),TILTCD
  451 BLDG=(TILT/2)*VALBDG/100
      GO TO 454
  452 IF(TILT.LE.20) GO TO 453
      BLDG=(10+(TILT-20))*VALBDG/100
      GO TO 454
  453 BLDG=(TILT/2)*VALBDG/100
  454 IF(CICODE.EQ.1) GO TO 455
      COALVL=CPTCUG*TTCOAL
      GO TO 456
  455 COALVL=CPTBFL*TTBFIL
  456 SCOCOS=RIVAGP+BLDG+COALVL
  460 WRITE(*,403) SCOCOS
  403 FORMAT('                    SUBSIDENCE COMPENSATION COST     $',I10)
      RETURN
      END
```

APPENDIX C

INPUT AND OUTPUT FORMAT EXAMPLE

INPUT DATA FOR FILE = 'COMDATA'

```
    0.0      .000
  2 4444  300  500    0
0.000  0.000
```

INPUT DATA FOR FILE = 'PILDATA'

```
  1100 150000     20   1986      0
     4.00     20.00     1.80     0.00     0.46    -0.66     1.50
1320.0
1330.0
1340.0
   0.0
```

INPUT DATA FOR FILE = 'USEDATA'

```
1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0 1.0
400000
 5.0
```

INPUT DATA FOR FILE = 'SUBSIDAT'

1	0	0	0	0	0	0

OUTPUT OF COMPRESSIVE STRENGTH GENERATOR

```
THE COMPRESSIVE STRENGTH IS =   479
THE COMPRESSIVE STRENGTH IS =   358
THE COMPRESSIVE STRENGTH IS =   307
THE COMPRESSIVE STRENGTH IS =   349
THE COMPRESSIVE STRENGTH IS =   446
THE COMPRESSIVE STRENGTH IS =   328
THE COMPRESSIVE STRENGTH IS =   499
THE COMPRESSIVE STRENGTH IS =   494
THE COMPRESSIVE STRENGTH IS =   464
THE COMPRESSIVE STRENGTH IS =   448
THE COMPRESSIVE STRENGTH IS =   341
THE COMPRESSIVE STRENGTH IS =   390
THE COMPRESSIVE STRENGTH IS =   331
THE COMPRESSIVE STRENGTH IS =   319
THE COMPRESSIVE STRENGTH IS =   433
THE COMPRESSIVE STRENGTH IS =   435
THE COMPRESSIVE STRENGTH IS =   448
THE COMPRESSIVE STRENGTH IS =   339
THE COMPRESSIVE STRENGTH IS =   375
THE COMPRESSIVE STRENGTH IS =   430
THE COMPRESSIVE STRENGTH IS =   415
THE COMPRESSIVE STRENGTH IS =   309
THE COMPRESSIVE STRENGTH IS =   368
THE COMPRESSIVE STRENGTH IS =   376
THE COMPRESSIVE STRENGTH IS =   432
THE COMPRESSIVE STRENGTH IS =   430
THE COMPRESSIVE STRENGTH IS =   410
THE COMPRESSIVE STRENGTH IS =   471
THE COMPRESSIVE STRENGTH IS =   300
THE COMPRESSIVE STRENGTH IS =   300
Stop - Program terminated.
```

SAFE PILLAR DIMENSION IS 68 FEET
PRIMARY MINING PERCENTAGE EXTRACTION = 40.29
THE PILLAR EXTRACTION METHOD USED FOR THIS PILLAR SIZE WILL BE "POCKET AND WIN
SECONDARY MINING PERCENTAGE EXTRACTION (MAX.) = 81.00
TOTAL TIMBER POSTS REQUIRED TO EXTRACT ONE PILLAR = 217
TOTAL TIMBER HEADBOARDS REQUIRED TO EXTRACT ONE PILLAR = 217
TOTAL NUMBER OF ROCKBOLTS REQUIRED TO EXTRACT ONE PILLAR = 190
TOTAL AMOUNT OF COAL RESERVE PER ACRE (VIRGIN PROPERTY) IN TONS = 7200
TONS MINED PER ACRE FROM PRIMARY MINING = 2900.83
TOTAL ACREAGE REQUIRED FOR PRIMARY MINING = 1034.19
EXTRA COAL MINED DURING SECONDARY MINING = 3031384
TOTAL COAL MINED BY EXTRACTING ONE PILLAR = 542.3
TOTAL NUMBER OF PILLARS TO BE EXTRACTED = 5590
EXTENDED LIFE OF MINE IN YEARS = *****
EXTENDED LIFE OF MINE IN MONTHS = 161.67
TOTAL LIFE OF MINE IN YEARS (INCLUDING SECONDARY MINING) = 33.473

ESTIMATED TOTAL PRODUCTION COST

DIRECT COST:	
Production & Maintenance:	
Labor and Supervision	$ 12735854
Operating Supplies:	
Rockbolts and Timbers	$ 9688557
Mining Machine Parts	$ 2334165
Lubrication & Hydraulic Oil	$ 927603
Rock Dust	$ 501188
Ventilation	$ 721469
Bits	$ 454707
Cables	$ 222301
Miscellaneous	$ 553732
Power	$ 2423086
Water	$ 8083
Royalty	$ 606276
Payroll Overhead (40% of Payroll)	$ 5094341
Union Welfare	$ 3446346
Reclamation Fund	$ 303138
Licenses	$ 202092
INDIRECT COST:	
15% of Labor, Supervision, & Supplies	$ 4220936
TOTAL	$ 44443874
FIXED COST:	
Taxes and Insurance (2% of Mine Cost)	$ 888877
Depreciation	$ 3991456
GRAND TOTAL (Fixed, Direct, & Indirect)	$ 49324207
DEPLETION ALLOWANCE	$ 1052563
EXTRA OPERATING COST DUE TO THINNING OF SEAM (Deviation from base model of 6 feet thick seam)	$ 4715486

SUBSIDENCE COMPENSATION COST	$	0
FINAL TOTAL	$	55092256
INFLATION ALLOWANCE	$	34107233
OVERALL COST OF SECONDARY MINING	$	89199489

COST OF SECONDARY MINING (PILLAR EXTRAC) PER ACRE= $ 86250

COST OF PILLAR EXTRACTION PER TON OF COAL = $29.00

SAFE PILLAR DIMENSION IS 72 FEET
PRIMARY MINING PERCENTAGE EXTRACTION = 38.75
THE PILLAR EXTRACTION METHOD USED FOR THIS PILLAR SIZE WILL BE "POCKET AND WIN
SECONDARY MINING PERCENTAGE EXTRACTION (MAX.) = 81.00
TOTAL TIMBER POSTS REQUIRED TO EXTRACT ONE PILLAR = 217
TOTAL TIMBER HEADBOARDS REQUIRED TO EXTRACT ONE PILLAR = 217
TOTAL NUMBER OF ROCKBOLTS REQUIRED TO EXTRACT ONE PILLAR = 190
TOTAL AMOUNT OF COAL RESERVE PER ACRE (VIRGIN PROPERTY) IN TONS = 7200
TONS MINED PER ACRE FROM PRIMARY MINING = 2790.17
TOTAL ACREAGE REQUIRED FOR PRIMARY MINING = 1075.20
EXTRA COAL MINED DURING SECONDARY MINING = 3270584
TOTAL COAL MINED BY EXTRACTING ONE PILLAR = 615.0
TOTAL NUMBER OF PILLARS TO BE EXTRACTED = 5318
EXTENDED LIFE OF MINE IN YEARS = *****
EXTENDED LIFE OF MINE IN MONTHS = 174.43
TOTAL LIFE OF MINE IN YEARS (INCLUDING SECONDARY MINING) = 34.536

ESTIMATED TOTAL PRODUCTION COST

DIRECT COST:		
Production & Maintenance:		
Labor and Supervision	$	13740813
Operating Supplies:		
Rockbolts and Timbers	$	9343385
Mining Machine Parts	$	2518349
Lubrication & Hydraulic Oil	$	1000798
Rock Dust	$	540736
Ventilation	$	778398
Bits	$	490587
Cables	$	239842
Miscellaneous	$	597426
Power	$	2614286
Water	$	8721
Royalty	$	654116
Payroll Overhead (40% of Payroll)	$	5496325
Union Welfare	$	3718290
Reclamation Fund	$	327058
Licenses	$	218038
INDIRECT COST:		
15% of Labor,Supervision, & Supplies	$	4387550
TOTAL	$	46674718

FIXED COST:

Taxes and Insurance (2% of Mine Cost)	$	933494
Depreciation	$	4306414
GRAND TOTAL (Fixed, Direct, & Indirect)	$	51914626

DEPLETION ALLOWANCE	$	1135619
EXTRA OPERATING COST DUE TO THINNING OF SEAM (Deviation from base model of 6 feet thick seam)	$	5087575
SUBSIDENCE COMPENSATION COST	$	0
FINAL TOTAL	$	58137820
INFLATION ALLOWANCE	$	35992720
OVERALL COST OF SECONDARY MINING	$	94130540

COST OF SECONDARY MINING (PILLAR EXTRAC) PER ACRE= $ 87546

COST OF PILLAR EXTRACTION PER TON OF COAL = $28.00
SAFE PILLAR DIMENSION IS 76 FEET
PRIMARY MINING PERCENTAGE EXTRACTION = 37.33
THE PILLAR EXTRACTION METHOD USED FOR THIS PILLAR SIZE WILL BE "POCKET AND WIN
SECONDARY MINING PERCENTAGE EXTRACTION (MAX.) = 80.00
TOTAL TIMBER POSTS REQUIRED TO EXTRACT ONE PILLAR = 233
TOTAL TIMBER HEADBOARDS REQUIRED TO EXTRACT ONE PILLAR = 233
TOTAL NUMBER OF ROCKBOLTS REQUIRED TO EXTRACT ONE PILLAR = 205
TOTAL AMOUNT OF COAL RESERVE PER ACRE (VIRGIN PROPERTY) IN TONS = 7200
TONS MINED PER ACRE FROM PRIMARY MINING = 2687.50
TOTAL ACREAGE REQUIRED FOR PRIMARY MINING = 1116.28
EXTRA COAL MINED DURING SECONDARY MINING = 3429768
TOTAL COAL MINED BY EXTRACTING ONE PILLAR = 676.4
TOTAL NUMBER OF PILLARS TO BE EXTRACTED = 5070
EXTENDED LIFE OF MINE IN YEARS = *****
EXTENDED LIFE OF MINE IN MONTHS = 182.92
TOTAL LIFE OF MINE IN YEARS (INCLUDING SECONDARY MINING) = 35.243

ESTIMATED TOTAL PRODUCTION COST

DIRECT COST:

Production & Maintenance:		
Labor and Supervision	$	14409598
Operating Supplies:		
Rockbolts and Timbers	$	9601205
Mining Machine Parts	$	2640921
Lubrication & Hydraulic Oil	$	1049508
Rock Dust	$	567054
Ventilation	$	816284
Bits	$	514465
Cables	$	251516
Miscellaneous	$	626504
Power	$	2741527
Water	$	9146
Royalty	$	685953

Payroll Overhead (40% of Payroll)	$ 5763839
Union Welfare	$ 3899265
Reclamation Fund	$ 342976
Licenses	$ 228651

INDIRECT COST:

15% of Labor,Supervision, & Supplies	$ 4571558
TOTAL	$ 48719970

FIXED COST:

Taxes and Insurance (2% of Mine Cost)	$ 974399
Depreciation	$ 4516013
GRAND TOTAL (Fixed, Direct,& Indirect)	$ 54210382

DEPLETION ALLOWANCE $ 1190891

EXTRA OPERATING COST DUE TO THINNING OF SEAM
(Deviation from base model of 6 feet thick seam) $ 5335194
SUBSIDENCE COMPENSATION COST $ 0

FINAL TOTAL $ 60736467

INFLATION ALLOWANCE $ 37601525

OVERALL COST OF SECONDARY MINING $ 98337992

COST OF SECONDARY MINING (PILLAR EXTRAC) PER ACRE= $ 88094

COST OF PILLAR EXTRACTION PER TON OF COAL = $28.00

NEW OUTPUT CORRESPONDING TO NEXT STRENGTH DATA
SAFE PILLAR DIMENSION IS 67 FEET
PRIMARY MINING PERCENTAGE EXTRACTION = 40.69
THE PILLAR EXTRACTION METHOD USED FOR THIS PILLAR SIZE WILL BE "POCKET AND WIN
SECONDARY MINING PERCENTAGE EXTRACTION (MAX.) = 81.00
TOTAL TIMBER POSTS REQUIRED TO EXTRACT ONE PILLAR = 217
TOTAL TIMBER HEADBOARDS REQUIRED TO EXTRACT ONE PILLAR = 217
TOTAL NUMBER OF ROCKBOLTS REQUIRED TO EXTRACT ONE PILLAR = 190
TOTAL AMOUNT OF COAL RESERVE PER ACRE (VIRGIN PROPERTY) IN TONS = 7200
TONS MINED PER ACRE FROM PRIMARY MINING = 2929.85
TOTAL ACREAGE REQUIRED FOR PRIMARY MINING = 1023.94
EXTRA COAL MINED DURING SECONDARY MINING = 2971646
TOTAL COAL MINED BY EXTRACTING ONE PILLAR = 524.8
TOTAL NUMBER OF PILLARS TO BE EXTRACTED = 5663
EXTENDED LIFE OF MINE IN YEARS = *****
EXTENDED LIFE OF MINE IN MONTHS = 158.49
TOTAL LIFE OF MINE IN YEARS (INCLUDING SECONDARY MINING) = 33.207

ESTIMATED TOTAL PRODUCTION COST

DIRECT COST:

Production & Maintenance:

Labor and Supervision	$ 12484875
Operating Supplies:	
Rockbolts and Timbers	$ 9782651
Mining Machine Parts	$ 2288167
Lubrication & Hydraulic Oil	$ 909323
Rock Dust	$ 491312
Ventilation	$ 707251
Bits	$ 445746
Cables	$ 217920
Miscellaneous	$ 542820
Power	$ 2375335
Water	$ 7924
Royalty	$ 594329
Payroll Overhead (40% of Payroll)	$ 4993950
Union Welfare	$ 3378431
Reclamation Fund	$ 297164
Licenses	$ 198109
INDIRECT COST:	
15% of Labor,Supervision, & Supplies	$ 4180509
TOTAL	$ 43895816
FIXED COST:	
Taxes and Insurance (2% of Mine Cost)	$ 877916
Depreciation	$ 3912799
GRAND TOTAL (Fixed, Direct,& Indirect)	$ 48686531
DEPLETION ALLOWANCE	$ 1031821
EXTRA OPERATING COST DUE TO THINNING OF SEAM (Deviation from base model of 6 feet thick seam)	$ 4622560
SUBSIDENCE COMPENSATION COST	$ 0
FINAL TOTAL	$ 54340912
INFLATION ALLOWANCE	$ 33642081
OVERALL COST OF SECONDARY MINING	$ 87982993

COST OF SECONDARY MINING (PILLAR EXTRAC) PER ACRE= $ 85925

COST OF PILLAR EXTRACTION PER TON OF COAL = $29.00
NEW OUTPUT CORRESPONDING TO NEXT STRENGTH DATA
SAFE PILLAR DIMENSION IS 67 FEET
PRIMARY MINING PERCENTAGE EXTRACTION = 40.69
THE PILLAR EXTRACTION METHOD USED FOR THIS PILLAR SIZE WILL BE "POCKET AND WIN
SECONDARY MINING PERCENTAGE EXTRACTION (MAX.) = 81.00
TOTAL TIMBER POSTS REQUIRED TO EXTRACT ONE PILLAR = 217
TOTAL TIMBER HEADBOARDS REQUIRED TO EXTRACT ONE PILLAR = 217
TOTAL NUMBER OF ROCKBOLTS REQUIRED TO EXTRACT ONE PILLAR = 190
TOTAL AMOUNT OF COAL RESERVE PER ACRE (VIRGIN PROPERTY) IN TONS = 7200
TONS MINED PER ACRE FROM PRIMARY MINING = 2929.85

TOTAL ACREAGE REQUIRED FOR PRIMARY MINING = 1023.94
EXTRA COAL MINED DURING SECONDARY MINING = 2971646
TOTAL COAL MINED BY EXTRACTING ONE PILLAR = 524.8
TOTAL NUMBER OF PILLARS TO BE EXTRACTED = 5663
EXTENDED LIFE OF MINE IN YEARS = *****
EXTENDED LIFE OF MINE IN MONTHS = 158.49
TOTAL LIFE OF MINE IN YEARS (INCLUDING SECONDARY MINING) = 33.207

ESTIMATED TOTAL PRODUCTION COST

DIRECT COST:		
Production & Maintenance:		
Labor and Supervision	$	12484875
Operating Supplies:		
Rockbolts and Timbers	$	9782651
Mining Machine Parts	$	2288167
Lubrication & Hydraulic Oil	$	909323
Rock Dust	$	491312
Ventilation	$	707251
Bits	$	445746
Cables	$	217920
Miscellaneous	$	542820
Power	$	2375335
Water	$	7924
Royalty	$	594329
Payroll Overhead (40% of Payroll)	$	4993950
Union Welfare	$	3378431
Reclamation Fund	$	297164
Licenses	$	198109
INDIRECT COST:		
15% of Labor,Supervision, & Supplies	$	4180509
TOTAL	$	43895816
FIXED COST:		
Taxes and Insurance (2% of Mine Cost)	$	877916
Depreciation	$	3912799
GRAND TOTAL (Fixed, Direct,& Indirect)	$	48686531
DEPLETION ALLOWANCE	$	1031821
EXTRA OPERATING COST DUE TO THINNING OF SEAM (Deviation from base model of 6 feet thick seam)	$	4622560
SUBSIDENCE COMPENSATION COST	$	0
FINAL TOTAL	$	54340912
INFLATION ALLOWANCE	$	33642081
OVERALL COST OF SECONDARY MINING	$	87982993

```
        COST OF SECONDARY MINING (PILLAR EXTRAC) PER ACRE= $ 85925

        COST OF PILLAR EXTRACTION PER TON OF COAL = $29.00
Program terminated.
```

APPENDIX D

USER'S MANUAL OF THE COMPUTER PROGRAM

INPUT VARIABLES FOR PILCOST PROGRAM

Three types of data lines are required for computer analysis. All the common data are written in one line. The data values are placed in the respective line columns as shown.

Description of Input Data

First Data Line:- All the input data on this line are integer values.

Depth - Depth of coal seam below the surface is expressed in feet and is written column 1 through 7. A depth of 1100 feet is shown in the proper columns in input data file = 'PILDATA'.

Yearly Production - Columns 8 through 14 are reserved for the yearly coal production from development-only sections. A value of 150000 tons is shown in input data file = 'PILDATA'.

Mining Life - The value representing primary mining life occupies columns 15 through 21 and the unit is in years. The value 20 years is shown in input data file = 'PILDATA'.

Updating Year - The year for updating the total production cost from base year mid-1977, is placed in columns 22 through 28. A value of 1986 is shown in input data file = 'PILDATA'.

Code for Cost Data - The code value to choose type of cost data. If the user prefers to use his own unit cost parameters values the value of code = 1, otherwise code value = 0 for built-in cost data. This value is placed in column 35. A value of 0 is shown in input data file = 'PILDATA'.

Second Data Line:- All the input data on this line are real (with decimal) numbers.

Height - Height of coal seam or workable height is expressed in feet and is written in column 1 through 10, with the decimal point in column 8. A value of 4.00 feet is shown in input data file = 'PILDATA'.

Room Width - Room width in the panel is expressed in feet and columns 11 through 20, are reserved for this. The decimal point is in column 18. The value 20.00 is shown.

Desired Safety Factor - Columns 21 through 30 are reserved for this variable. The decimal point is in column 28. A value of 1.80 is shown in input data file = 'PILDATA'.

C - Coefficient 'C' in in pillar strength equation is written in columns 31 through 40 with the decimal point in column 38. A value of 0.00 is shown in input data file.

Alpha - Power index for width of pillar in pillar strength equation is written in columns 41 through 50 with the decimal point in column 48. A value of 0.46 shown in input data file = 'PILDATA'.

Beta - Power index for height of the pillar in pillar strength equation is written in columns 51 through 60 with the decimal point in column 58. A value of -0.66 for Salamon's equation is shown in input data file.

Ratio - Columns 61 through 70 are reserved for the ratio of secondary to primary mining productivities. The decimal point is put in column 68.

Third Data Line:- This is for coal strength value and it requires the strength value in real number format with columns 1 through 7 reserved for this. A decimal point is put in column 6. This format is repeated for every coal strength data. To end the input data file, the last line contains 0.0.

INPUT DATA FILE FOR SUBROUTINE SUBCOM.FOR

Two types of data lines are required for this subroutine. All the common data are written in one line.

The data values are placed in respective line columns as shown.

Description of Input Data

First Data Line:- All the input data on this line are integer values.

Code for Type of Surface Land - This is written in column 10.

Reduction in Value of Agricultural Property - Columns 11 through 20 are reserved for this value. A value of 0 is shown in input data file = 'SUBSIDAT'.

Construction Cost - Construction cost of building in 1913 occupies columns 21 through 30 written in dollars. The value 0 is shown in input data file.

Building Price Index - This is based on age of building and occupies columns 31 through 40. A value of 0 is shown in input data file = 'SUBSIDAT'.

Age of Building - Age of building is expressed in years and columns 41 through 50 are reserved for this. The value 0 is shown in input data file.

Tilt Code - Tilt code to choose one of the methods of computing loss in value of a building due to subsidence. Column 60 is reserved for this code value.

Tilt - Tilt value in mm/Metre. Columns 61 through 70 are reserved for this.

<u>Second Data Line</u> - This line has mixed (integer and real) numbers.

<u>Code</u> - Code to choose backfilling method or leaving coal pillars underground to protect surface installations. Column number 1 is reserved for this.

<u>Cost Per Ton of Coal Left Underground</u> - This is a real number and column 2 through 6 are reserved for this with decimal in column 4.

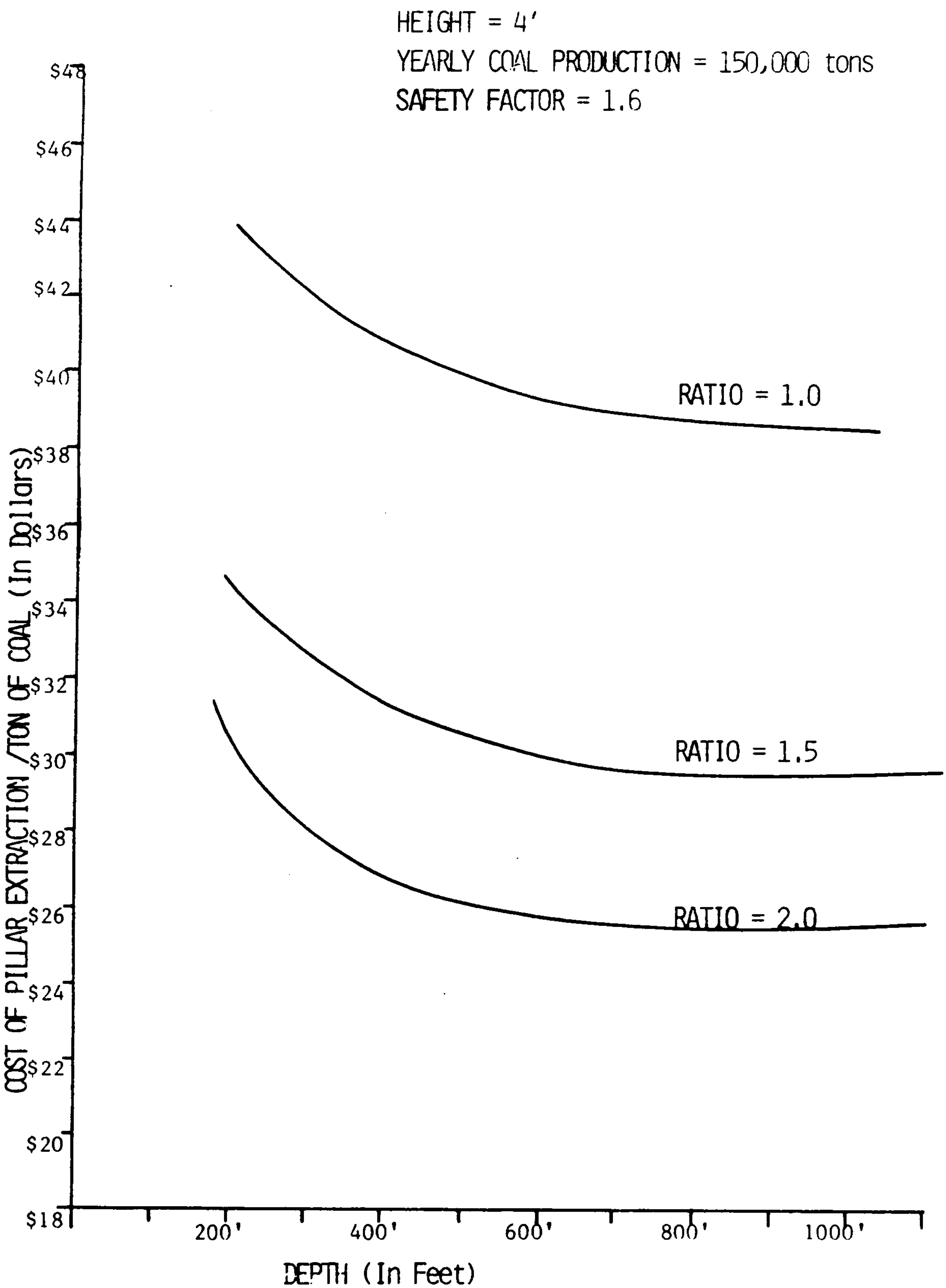
<u>Total Tons of Coal Left Underground</u> - Columns 7 through 16 are reserved for this. This is expressed as an integer.

<u>Cost Per Ton of Backfilling Required</u> - Columns 17 through 21 are reserved for this with the decimal point in column 19.

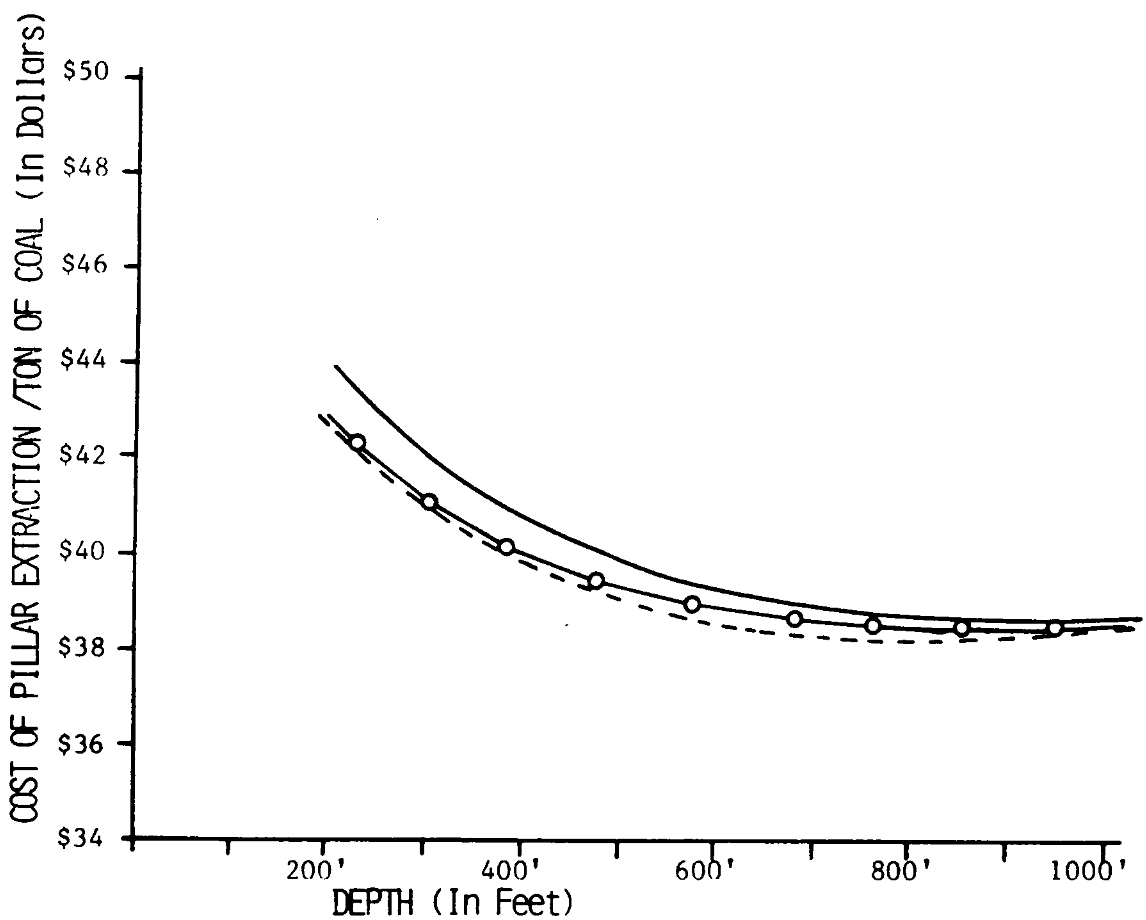
<u>Total Backfilling Required</u> - This is expressed as an integer and occupies columns 22 through 31.

APPENDIX E

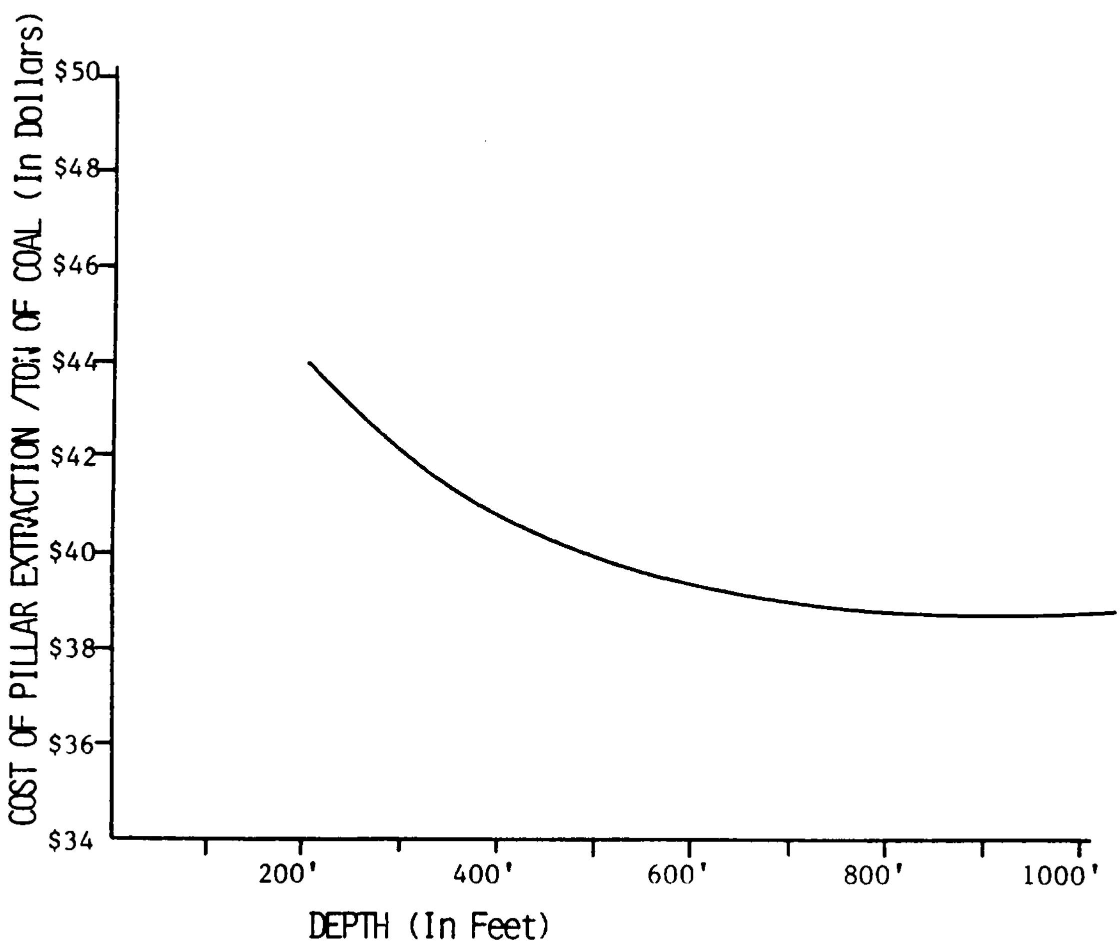
SENSITIVITY ANALYSIS RESULTS


HEIGHT = 4'
YEARLY COAL PRODUCTION = 150,000 tons
SAFETY FACTOR = 1.6
RATIO = 1.0
RATIO = 1.5
RATIO = 2.0
$48
$46
$44
$42
$40
$38
$36
$34
$32
$30
$28
$26
$24
$22
$20
$18
COST OF PILLAR EXTRACTION /TON OF COAL (In Dollars)
200'
400'
600'
800'
1000'
DEPTH (In Feet)

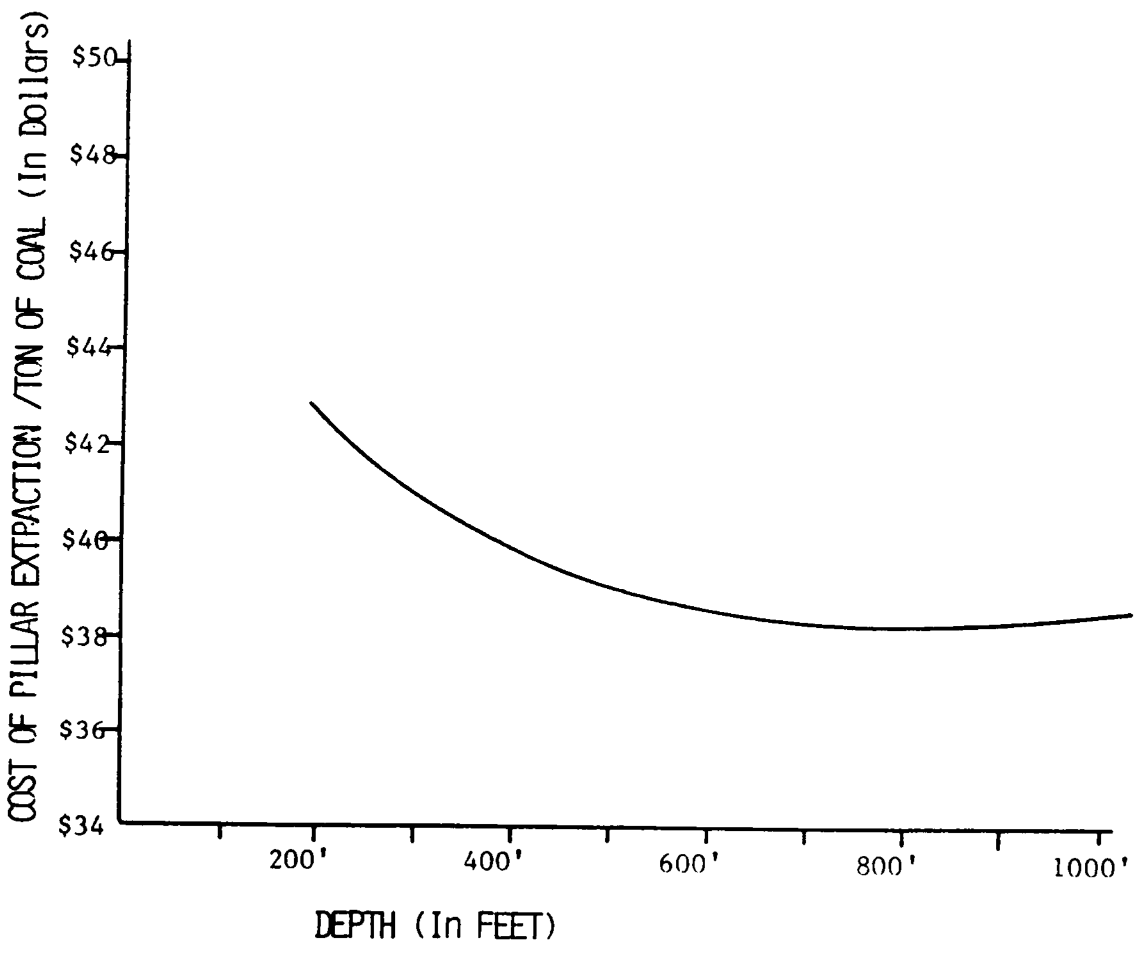
ENTRY WIDTH = 20′
SEAM THICKNESS = 4′
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION = 150,000 tons
MINING LIFE = 20 Years
YEAR OF PRODUCTION = 1986
RATIO = 1.0
SAFETY FACTOR = 1.6
= 1.8
= 2.0



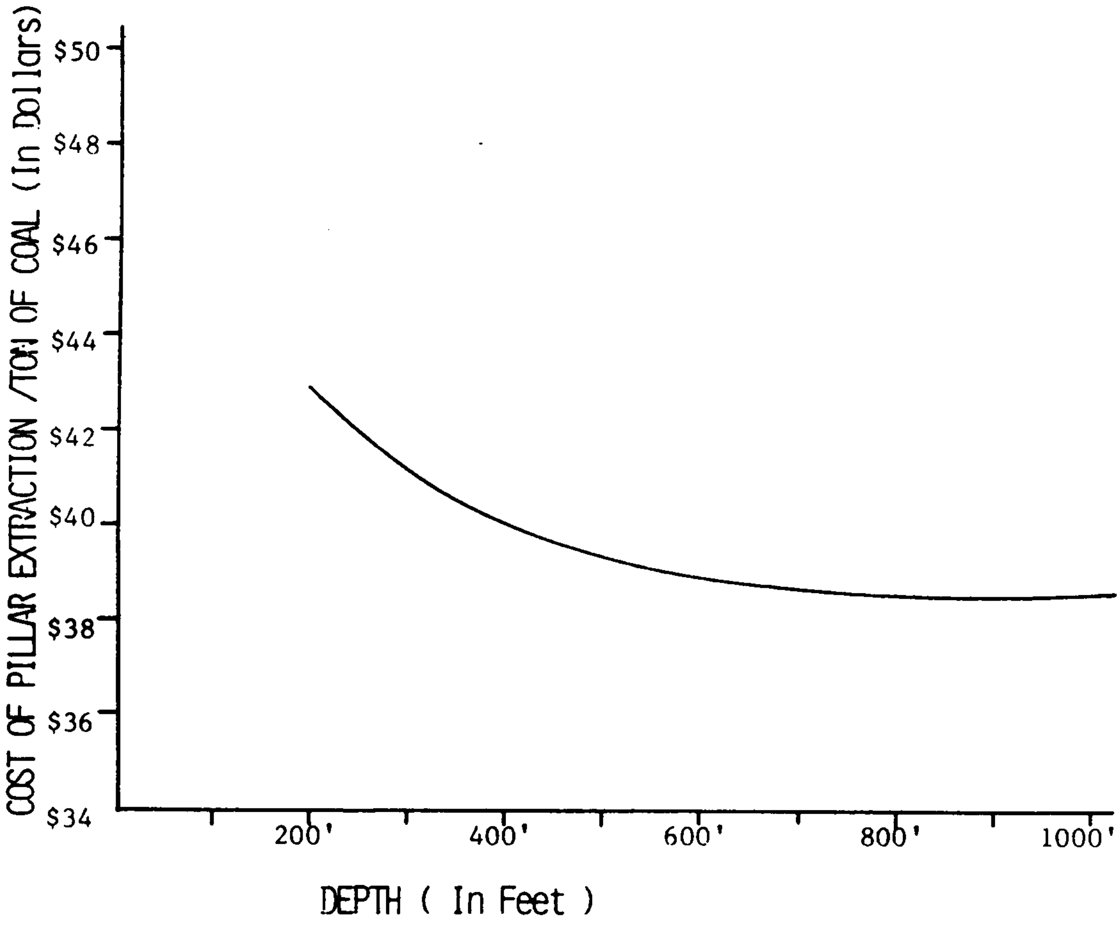
ENTRY WIDTH = 20'
SEAM THICKNESS = 4'
SAFETY FACTOR = 1.6 (Salamon's Equation)
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION = 150,000 tons (DEVELOPMENT)
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.00



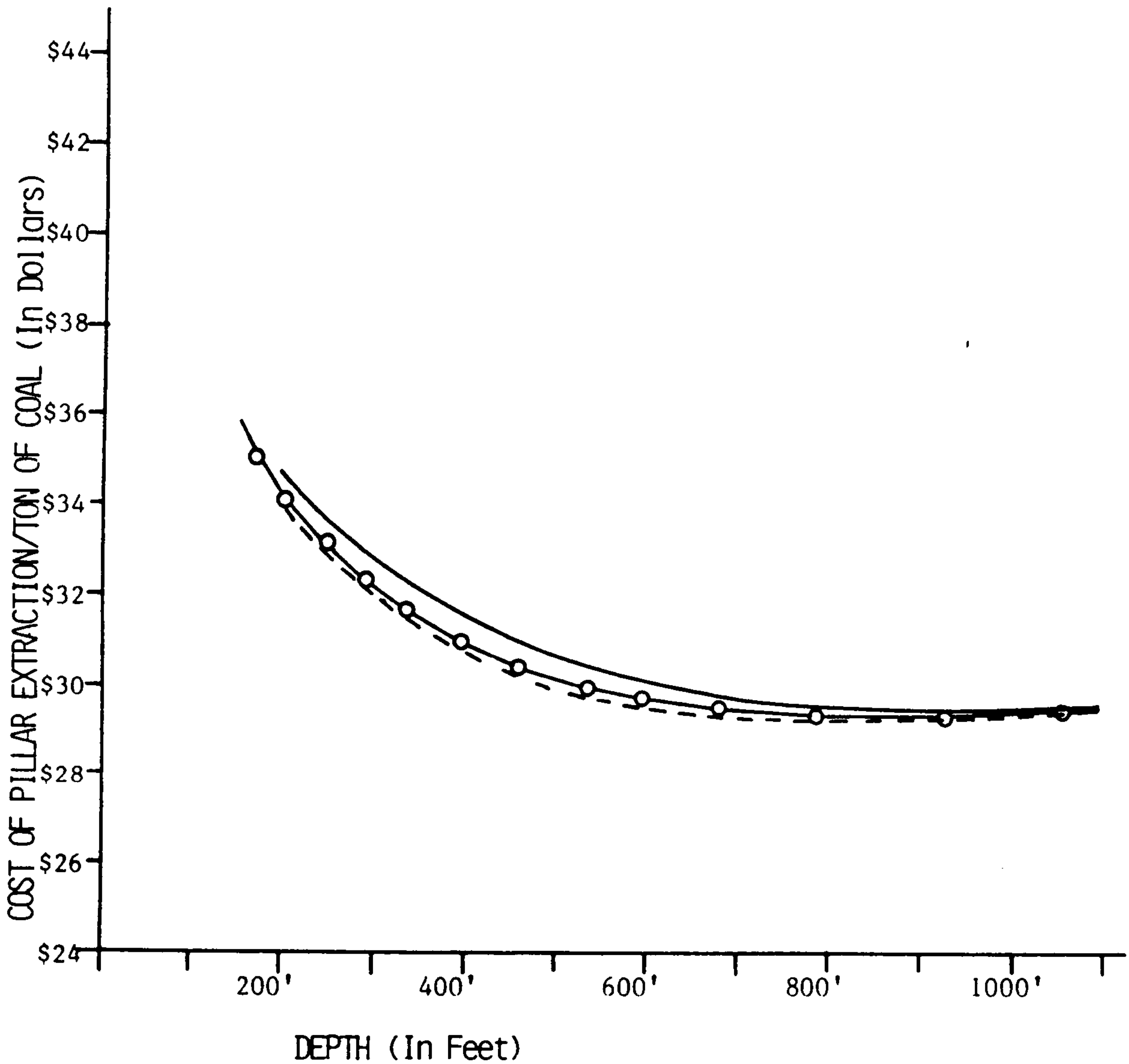
ENTRY WIDTH = 20'
SEAM THICKNESS = 4'
SAFETY FACTOR = 1.8 (Salamon's Equation)
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 150,000 Tons
MINING LIFE = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.00



ENTRY WIDTH = 20'
SEAM THICKNESS = 4'
SAFETY FACTOR = 2.0 (Salamon's Equation)
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 150,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.00



ENTRY WIDTH = 20'
SEAM THICKNESS = 4'
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 150,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.50
SAFETY FACTOR = 1.6
= 1.8
= 2.0



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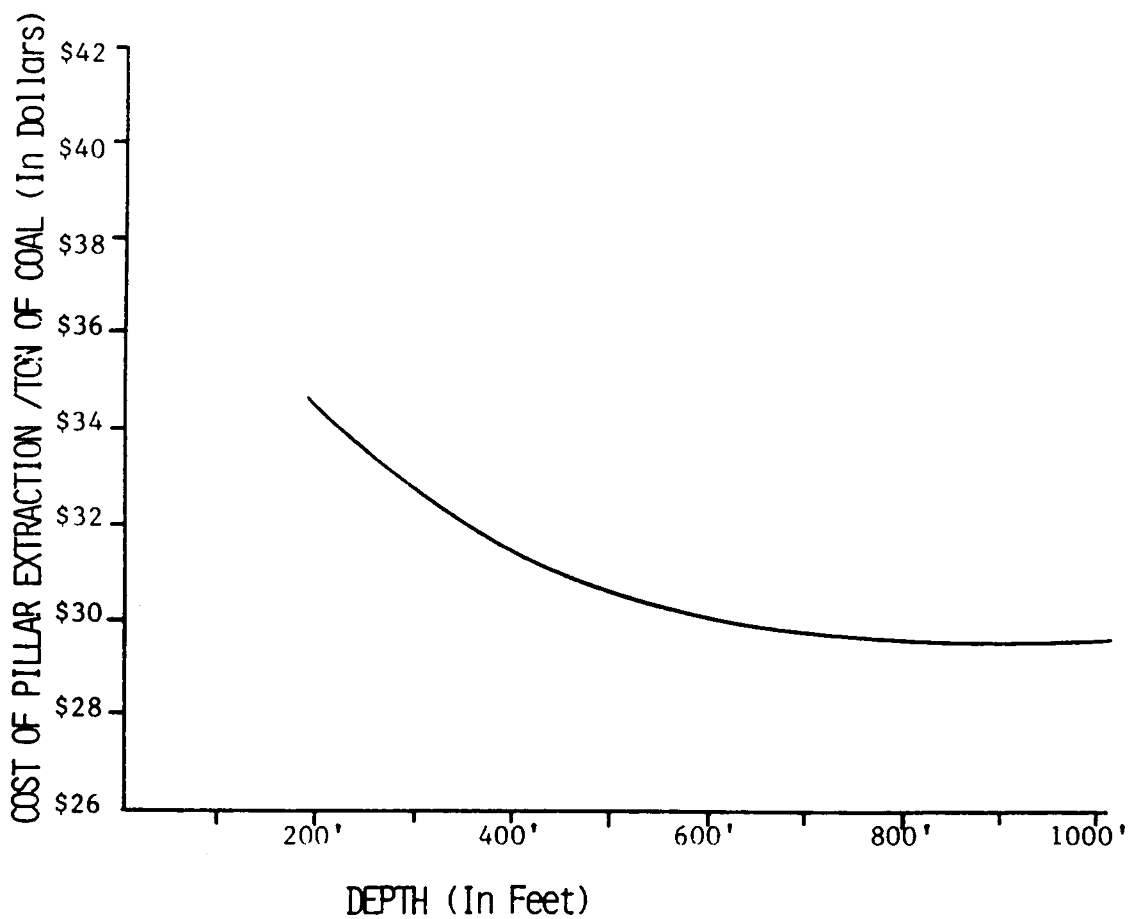
SEAM THICKNESS = 4'
SAFETY FACTOR = 1.6 (Salamon's Equation)
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 150,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.50



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SAFETY FACTOR = 1.8 (Salamon's Equation)
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
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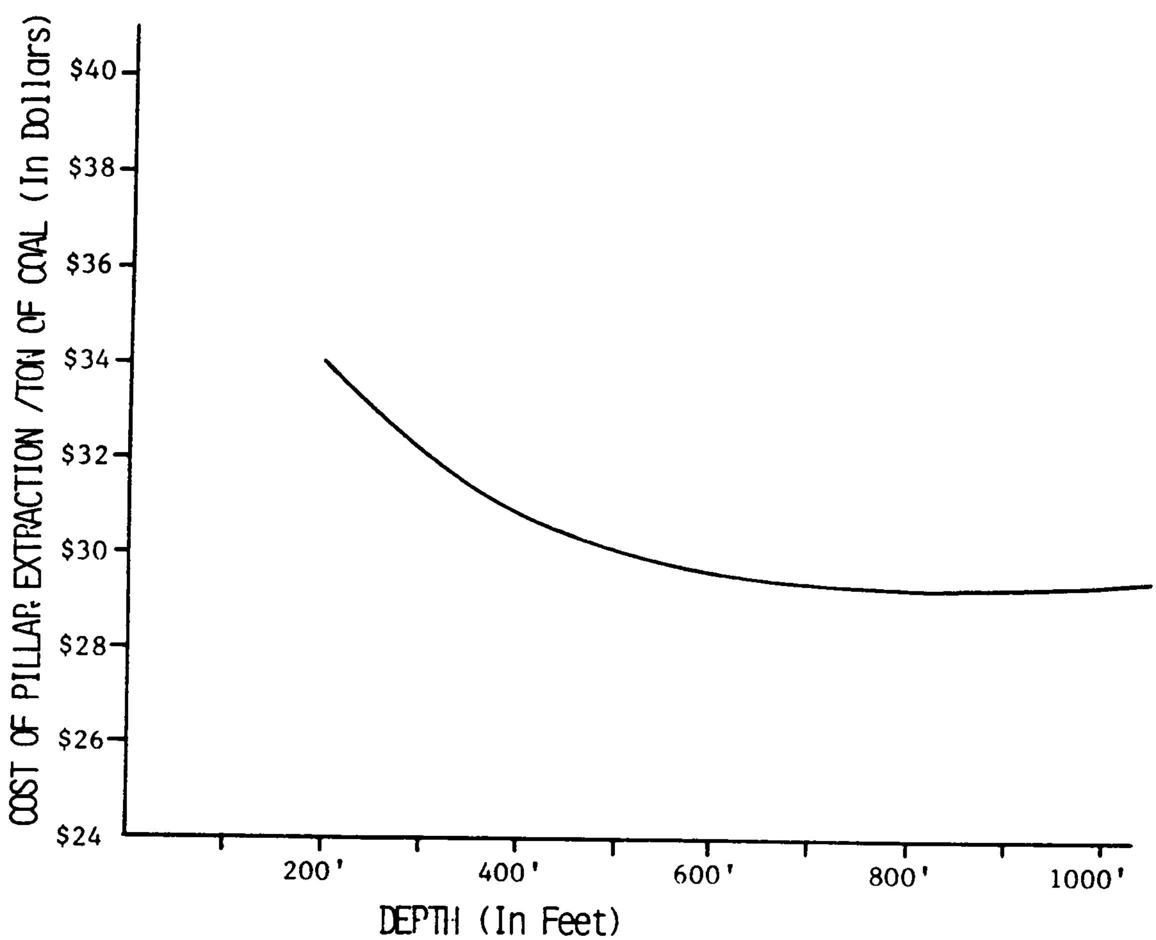
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.50



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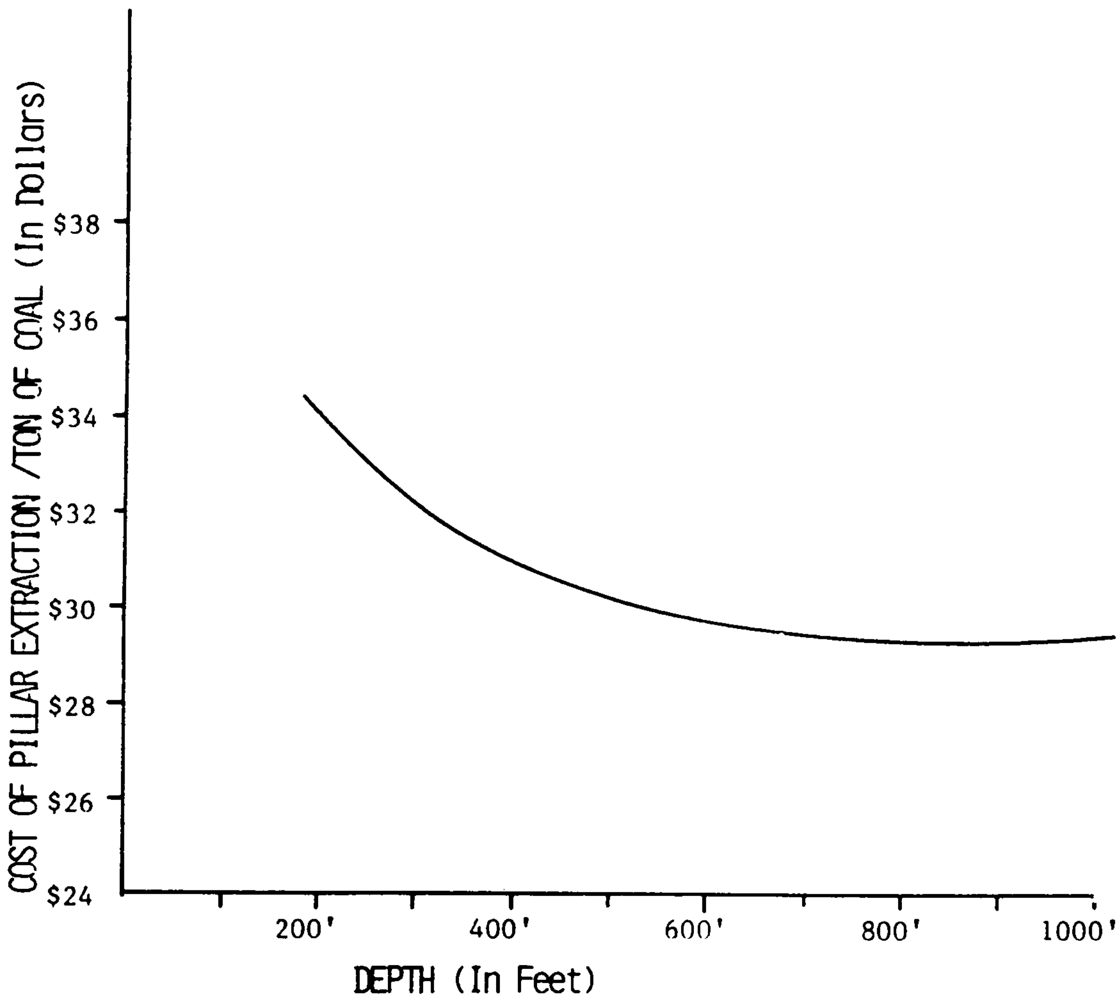
SAFETY FACTOR = 2.0 (Salamon's Equation)
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION = 150,000 Tons (DEVELOPMENT)
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.50



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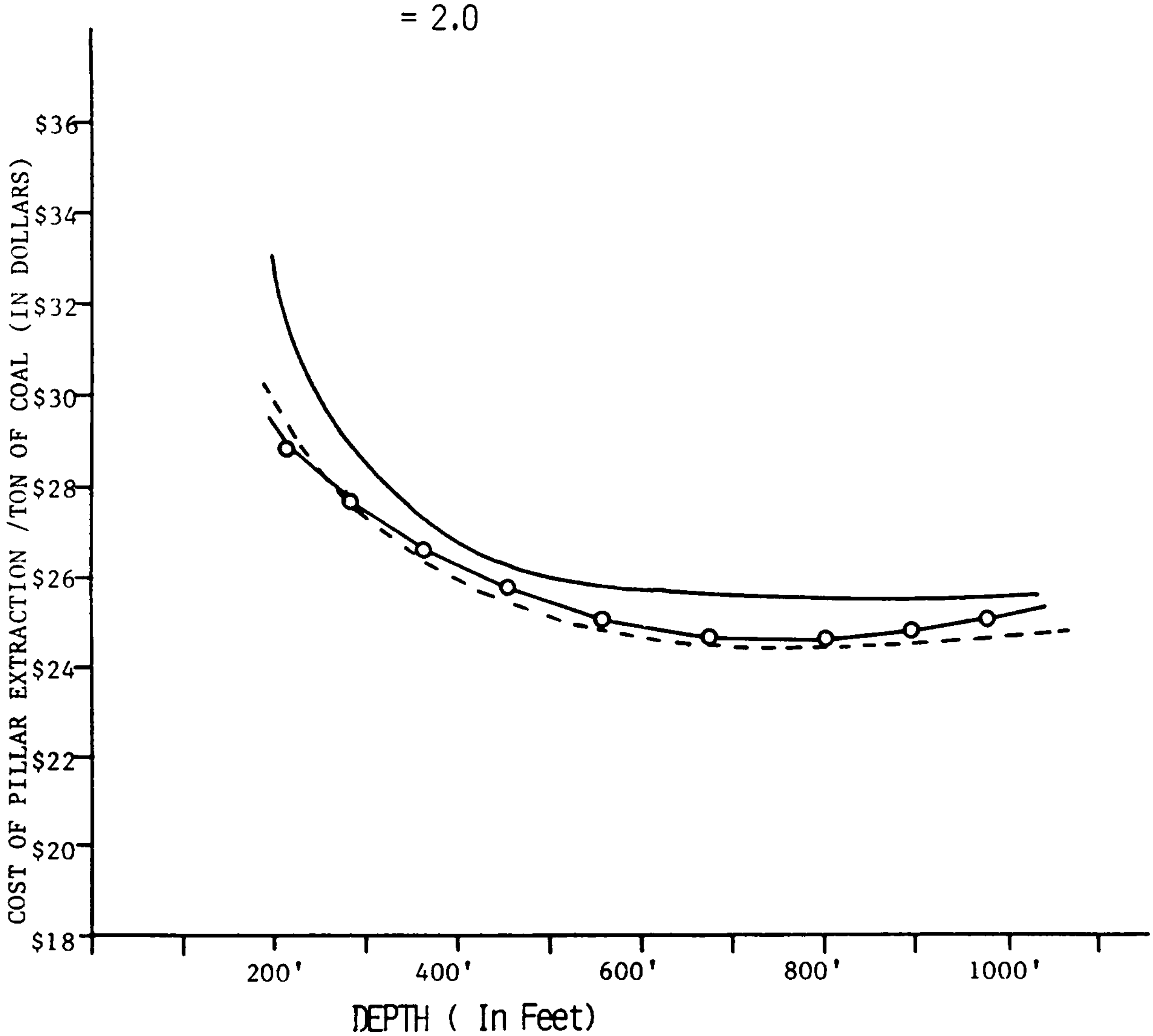
SEAM THICKNESS = 4′
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 150,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 2.00
SAFETY FACTOR = 1.6
= 1.8
= 2.0



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SUBSIDENCE COMPENSATION COST = 0 (Assumed)
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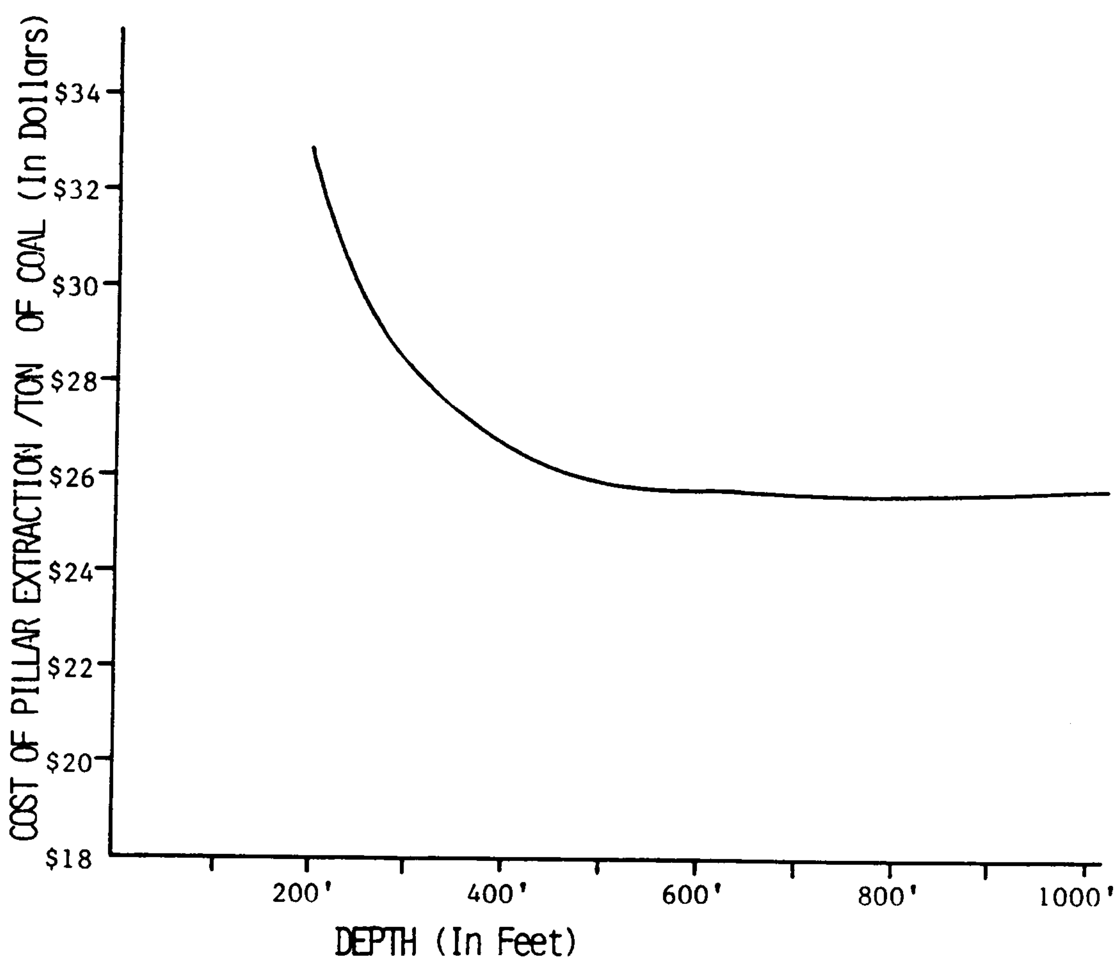
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 2.00



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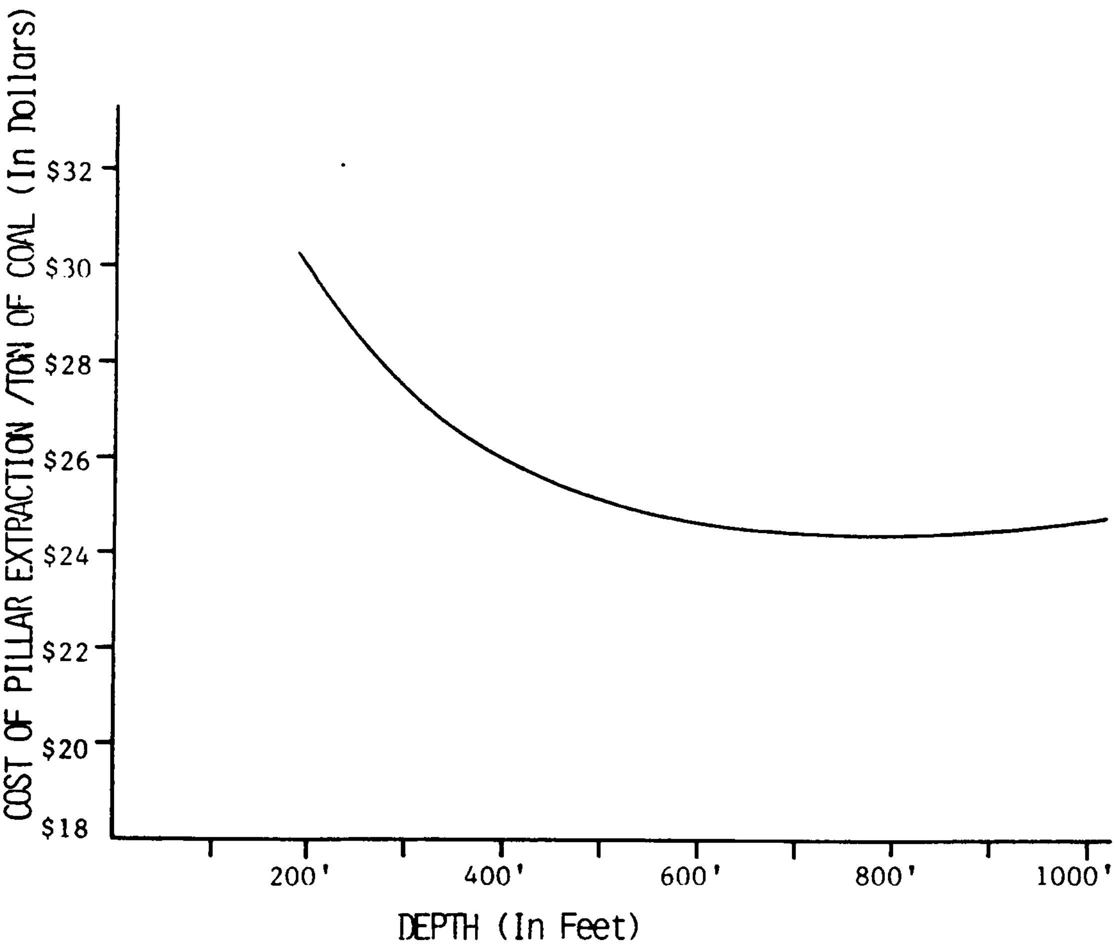
SAFETY FACTOR = 1.8 (Salamon's Equation)
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 150,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 2.00



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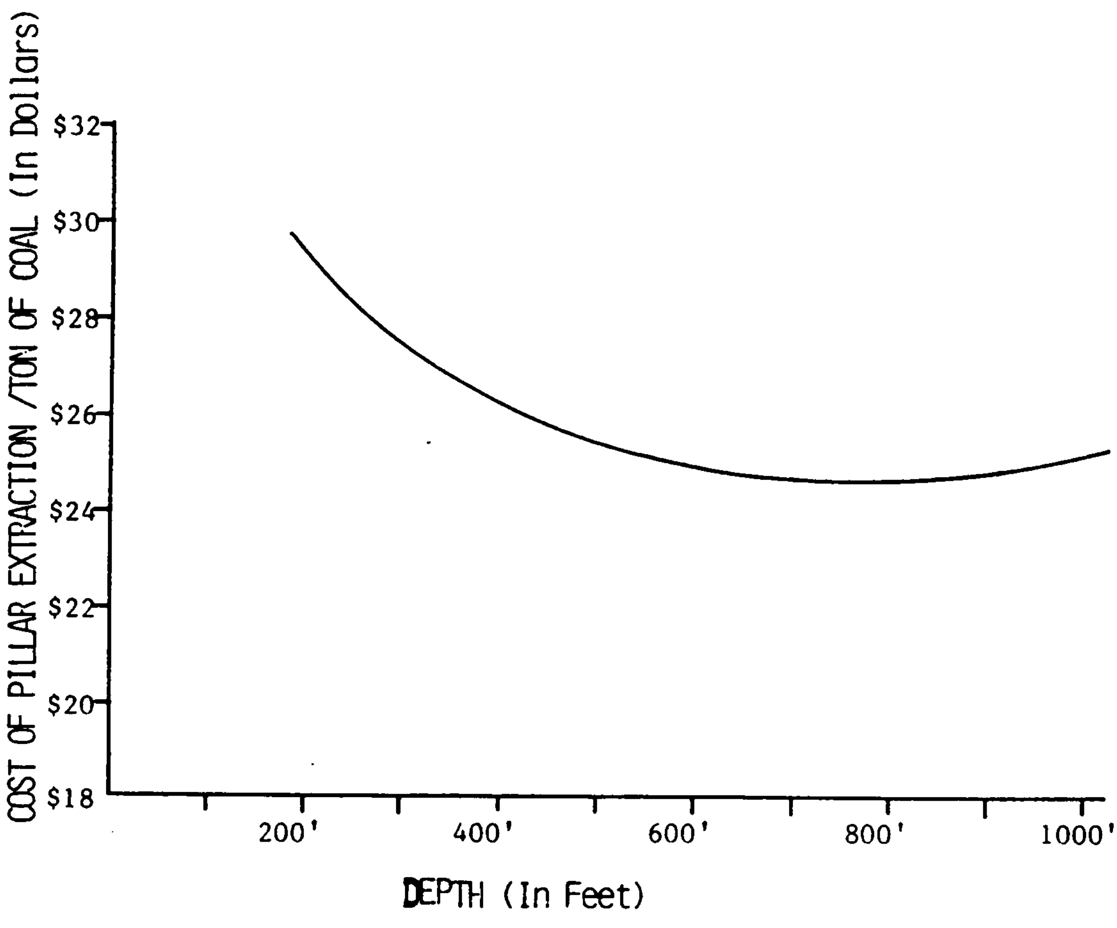
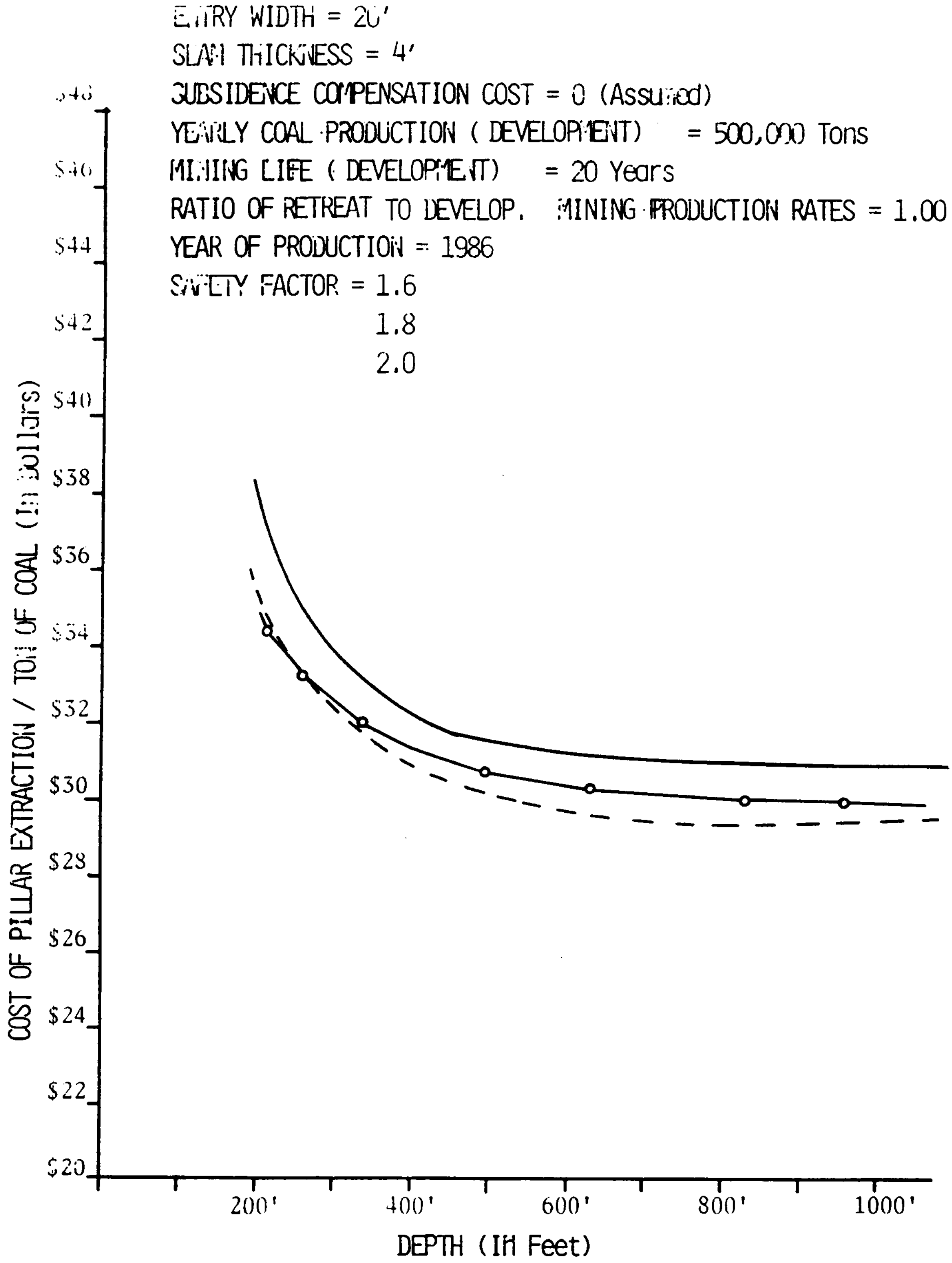
SAFETY FACTOR = 2.0 (Salamon's Equation)
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION = 150,000 Tons (DEVELOPMENT)
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 2.00

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SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 500,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.00
YEAR OF PRODUCTION = 1986
SAFETY FACTOR = 1.6
1.8
2.0
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
$46
$44
$42
$40
$38
$36
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$32
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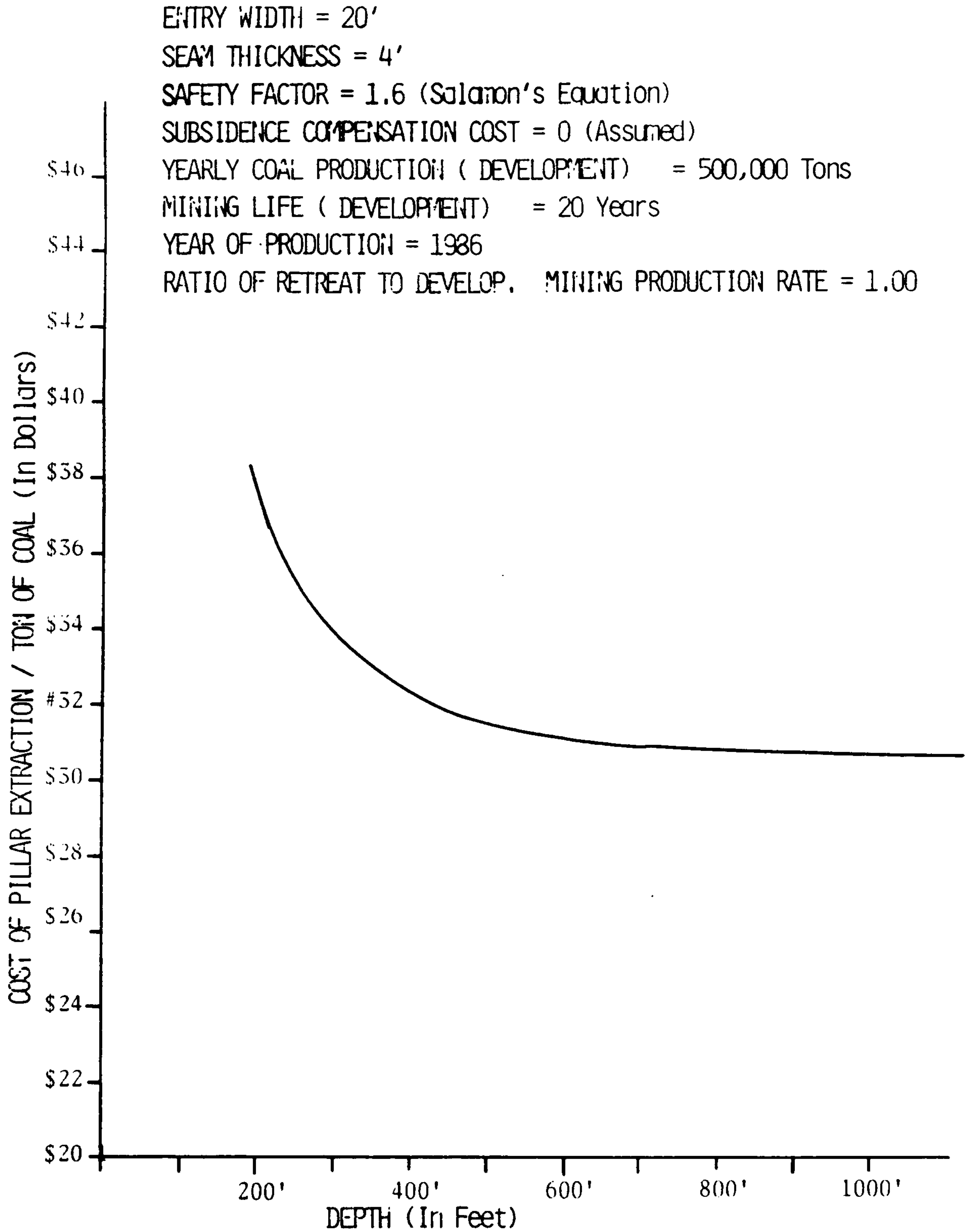
$26
$24
$22
$20
200'
400'
600'
800'
1000'
DEPTH (In Feet)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SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 500,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.00
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
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200'
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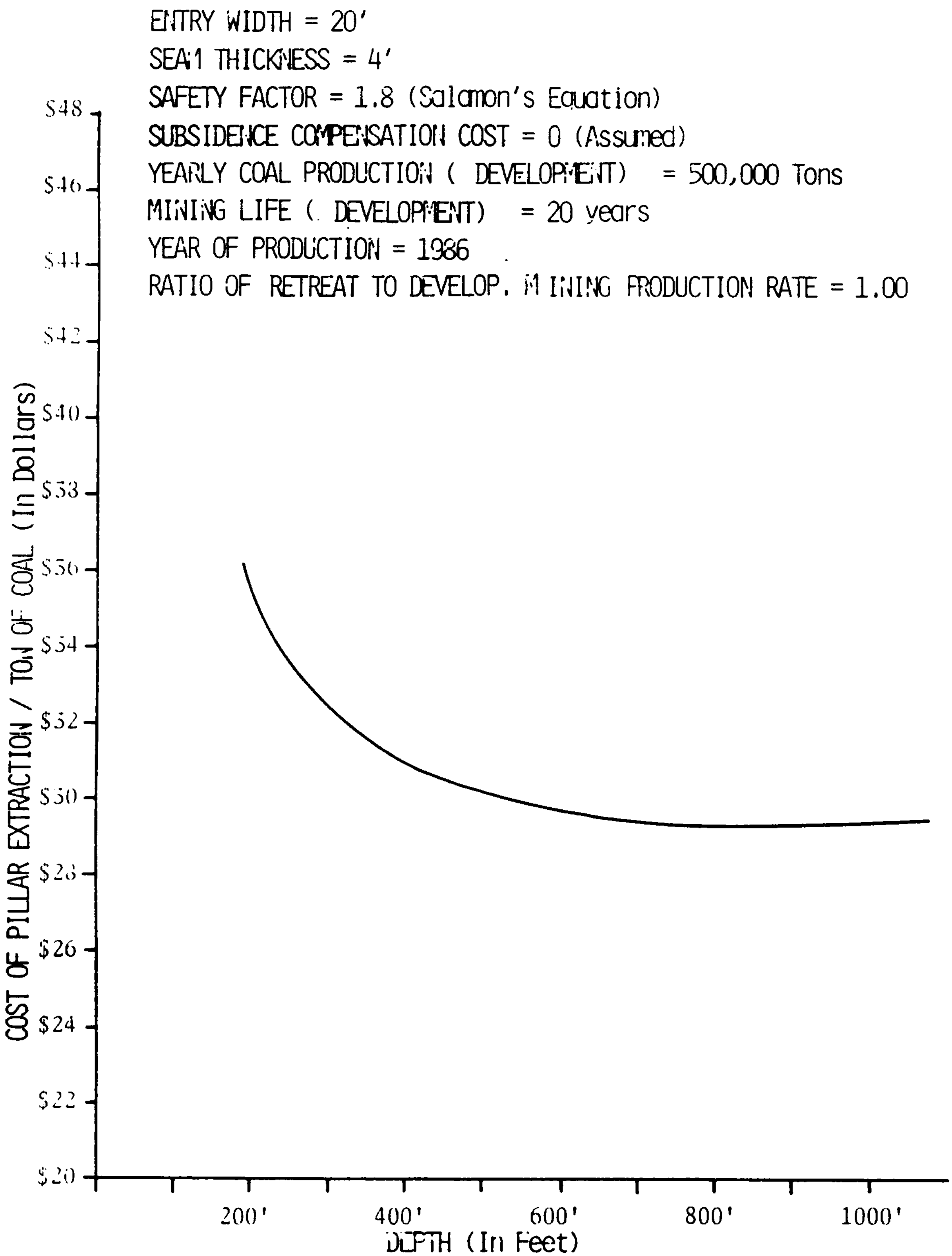
1000'
DEPTH (In Feet)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SAFETY FACTOR = 1.8 (Salamon's Equation)
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
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MINING LIFE (DEVELOPMENT) = 20 years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.00
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
$48
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$44
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200'
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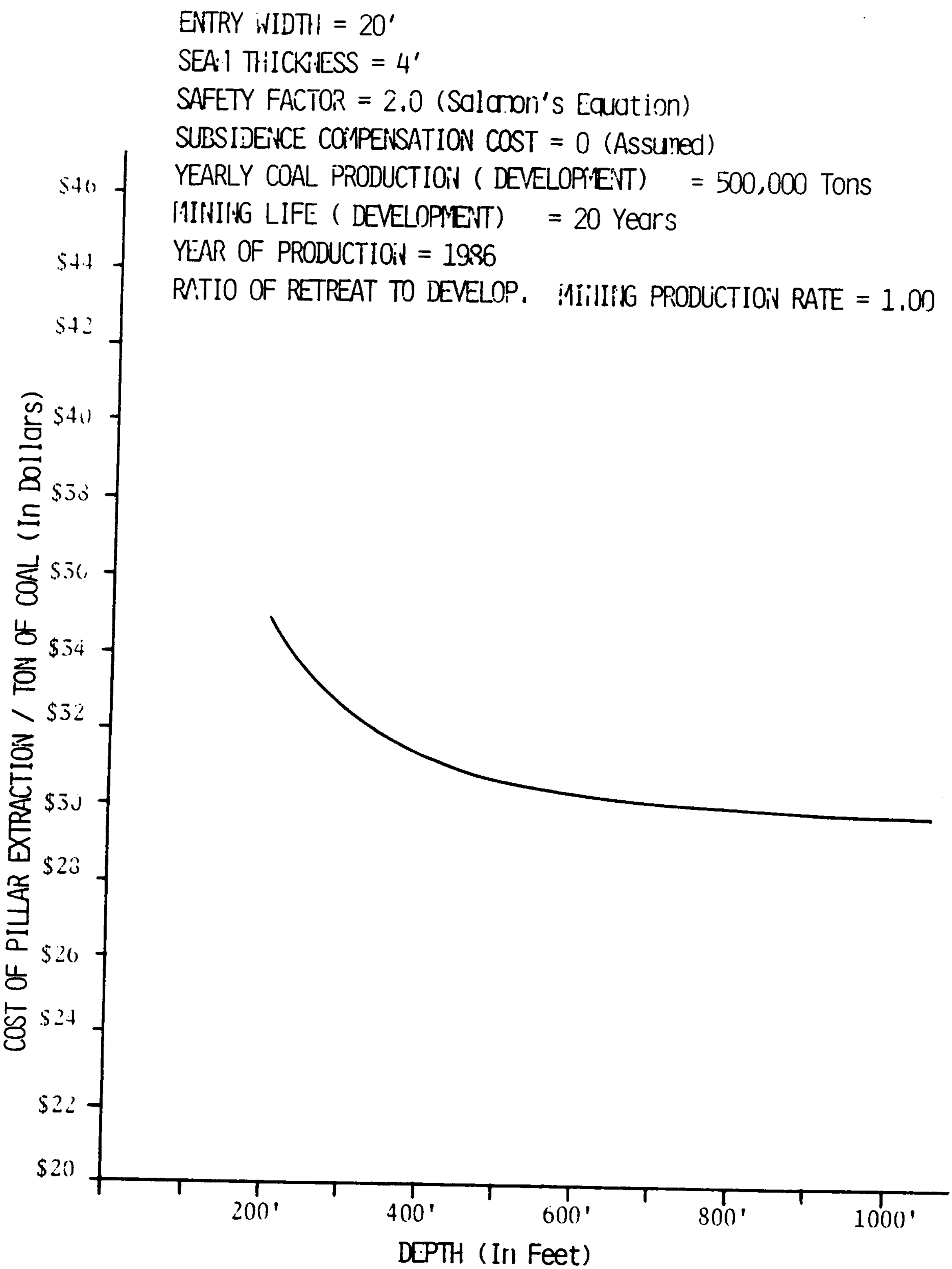
800'
1000'
DEPTH (In Feet)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SAFETY FACTOR = 2.0 (Salamon's Equation)
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 500,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.00
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
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200'
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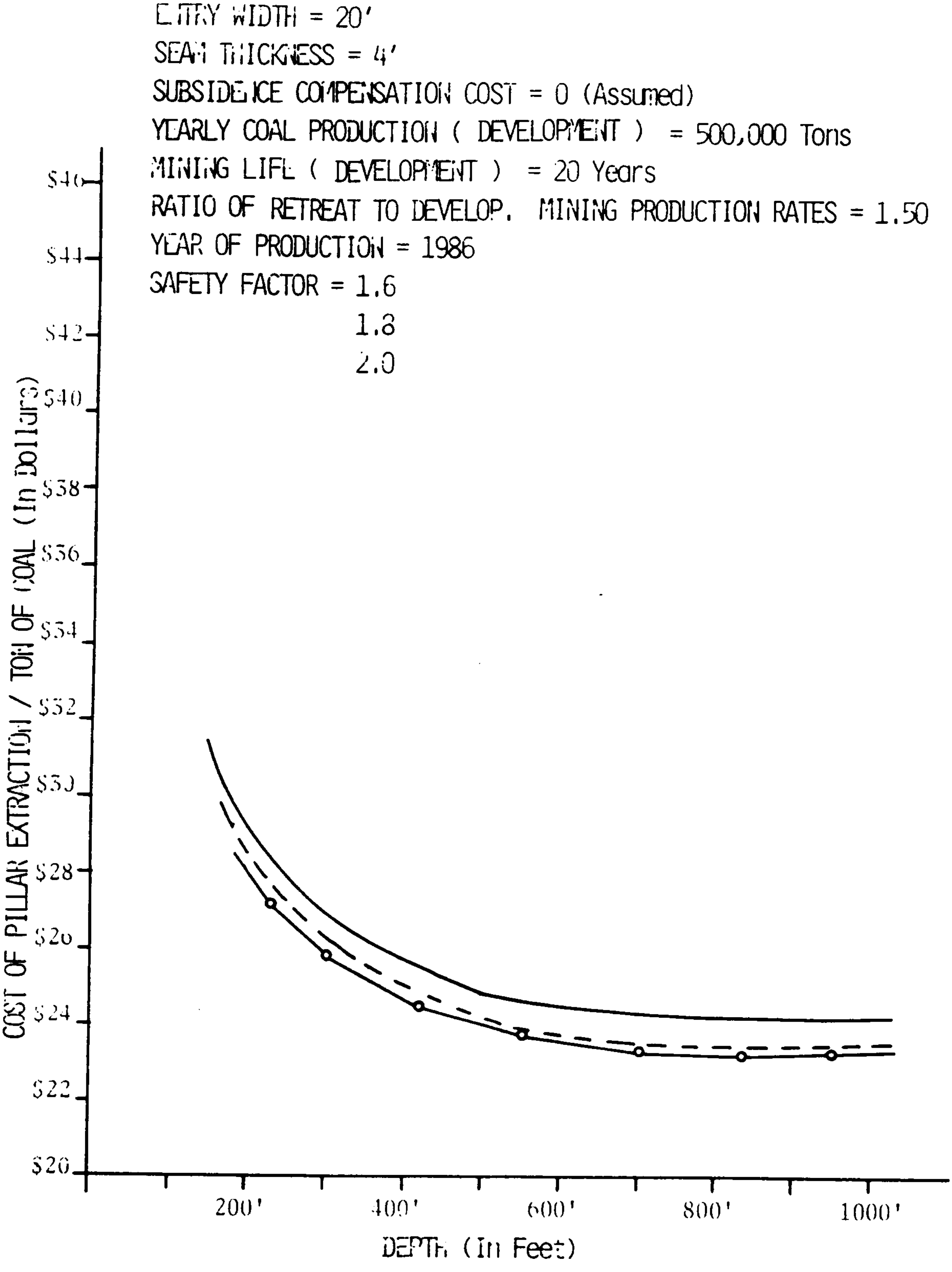
1000'
DEPTH (In Feet)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SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 500,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.50
YEAR OF PRODUCTION = 1986
SAFETY FACTOR = 1.6
1.8
2.0
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
$46
$44
$42
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200'
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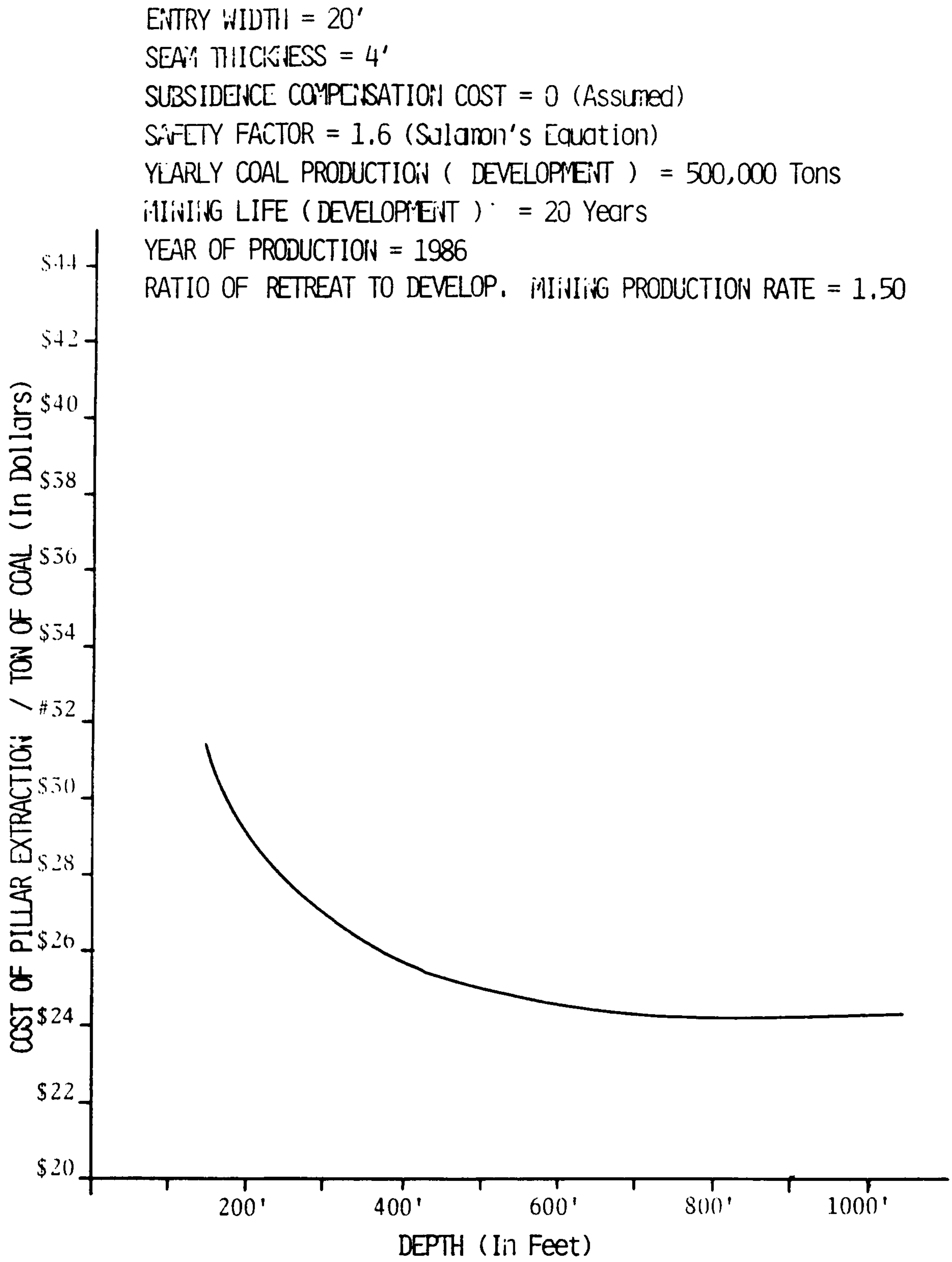
800'
1000'
DEPTH (In Feet)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SEAM THICKNESS = 4′
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
SAFETY FACTOR = 1.6 (Salamon's Equation)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 500,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.50
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
$44
$42
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#32
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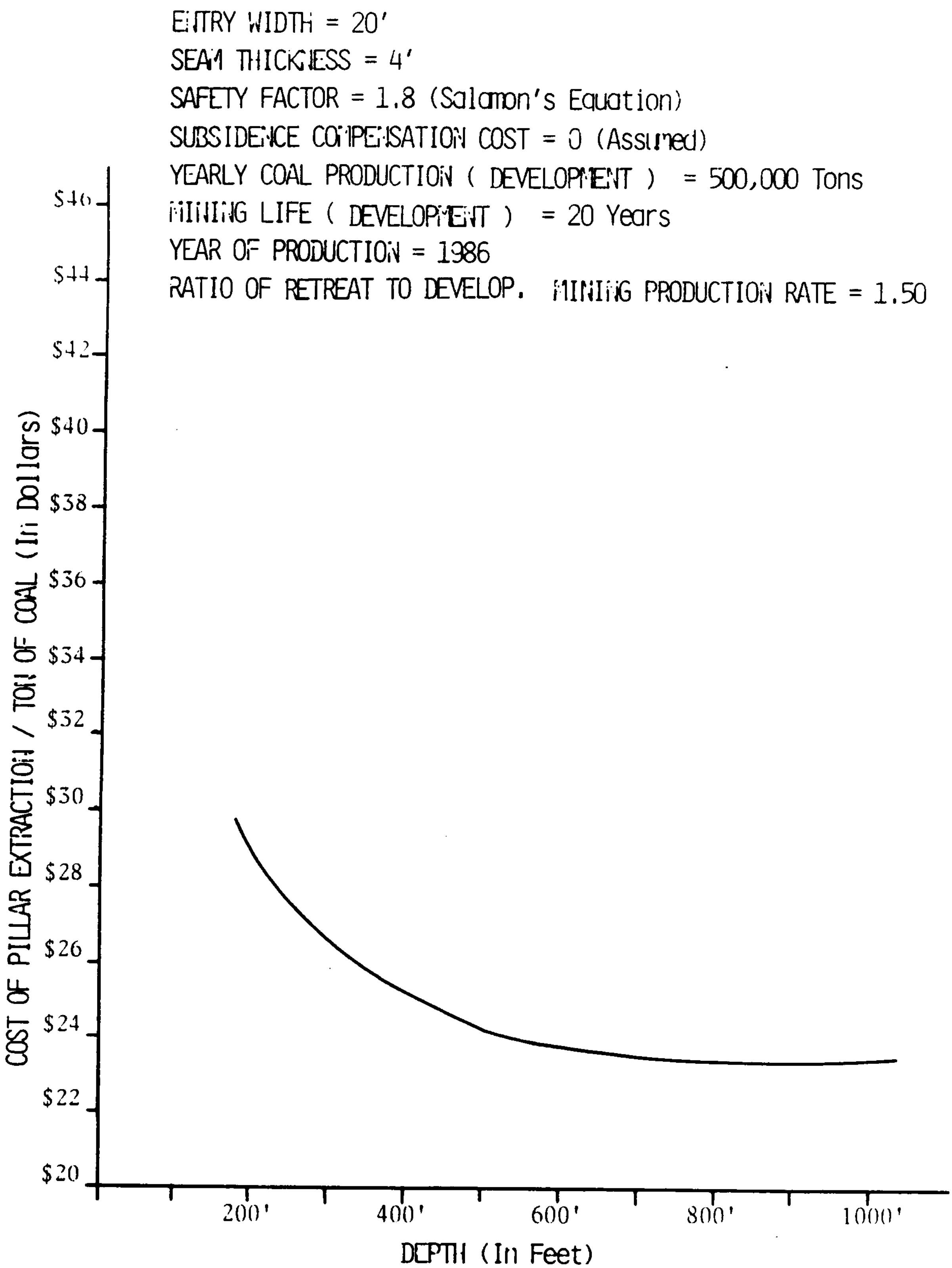
$20
200'
400'
600'
800'
1000'
DEPTH (In Feet)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SUBSIDENCE COMPENSATION COST = 0 (Assumed)
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MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.50
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
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200'
400'
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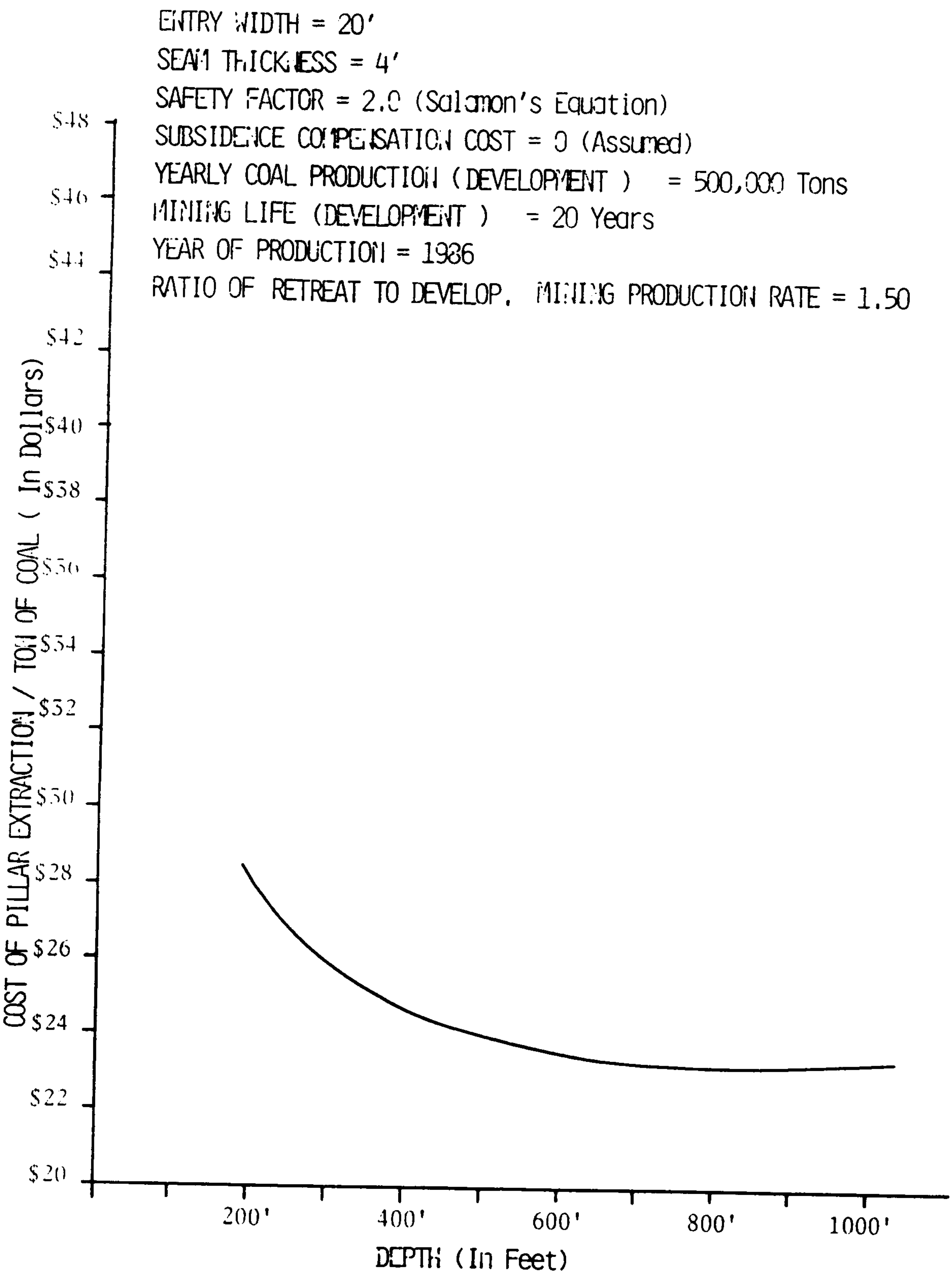
800'
1000'
DEPTH (In Feet)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MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.50
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
$48
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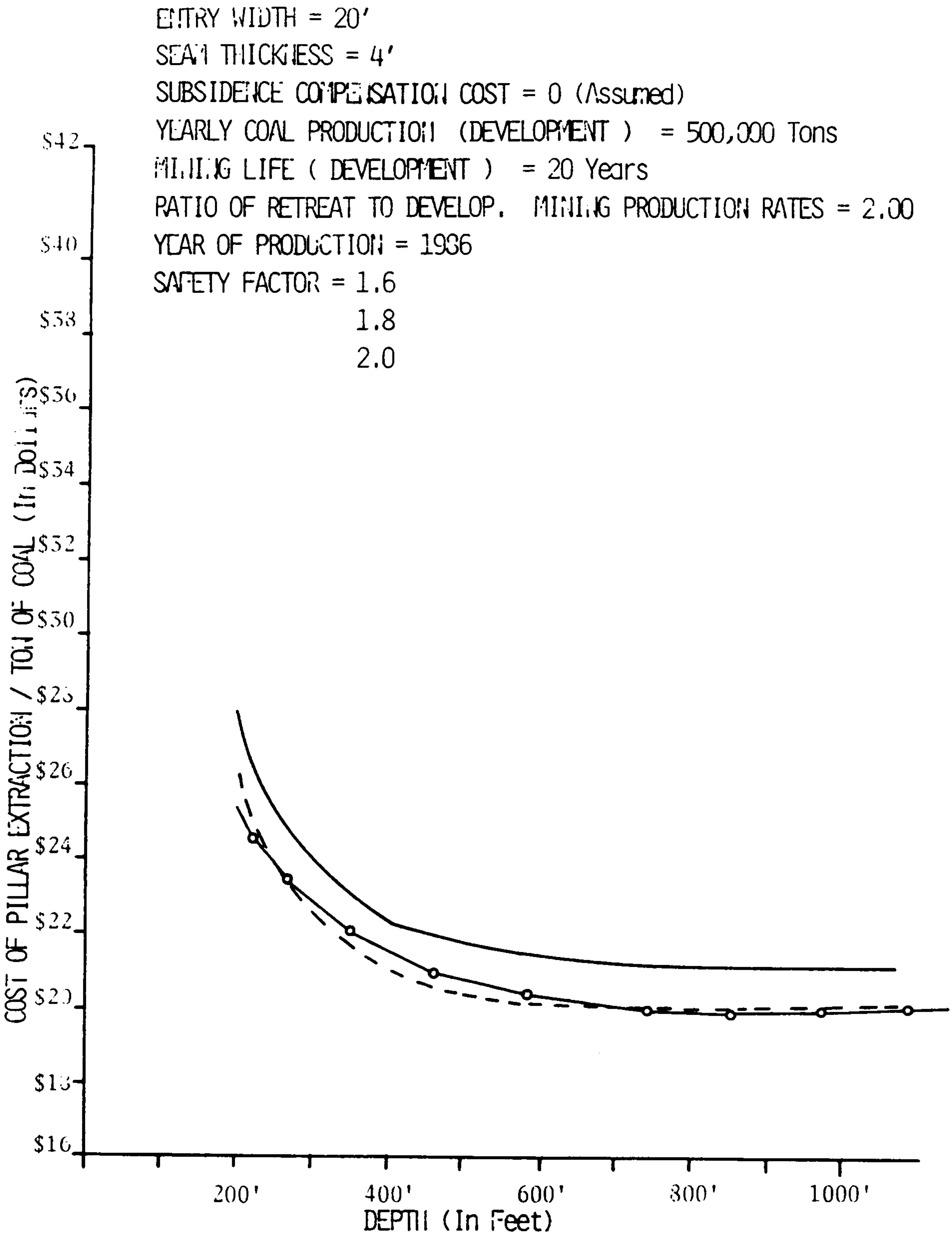
200'
400'
600'
800'
1000'
DEPTH (In Feet)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SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 500,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 2.00
YEAR OF PRODUCTION = 1986
SAFETY FACTOR = 1.6
1.8
2.0
$42
$40
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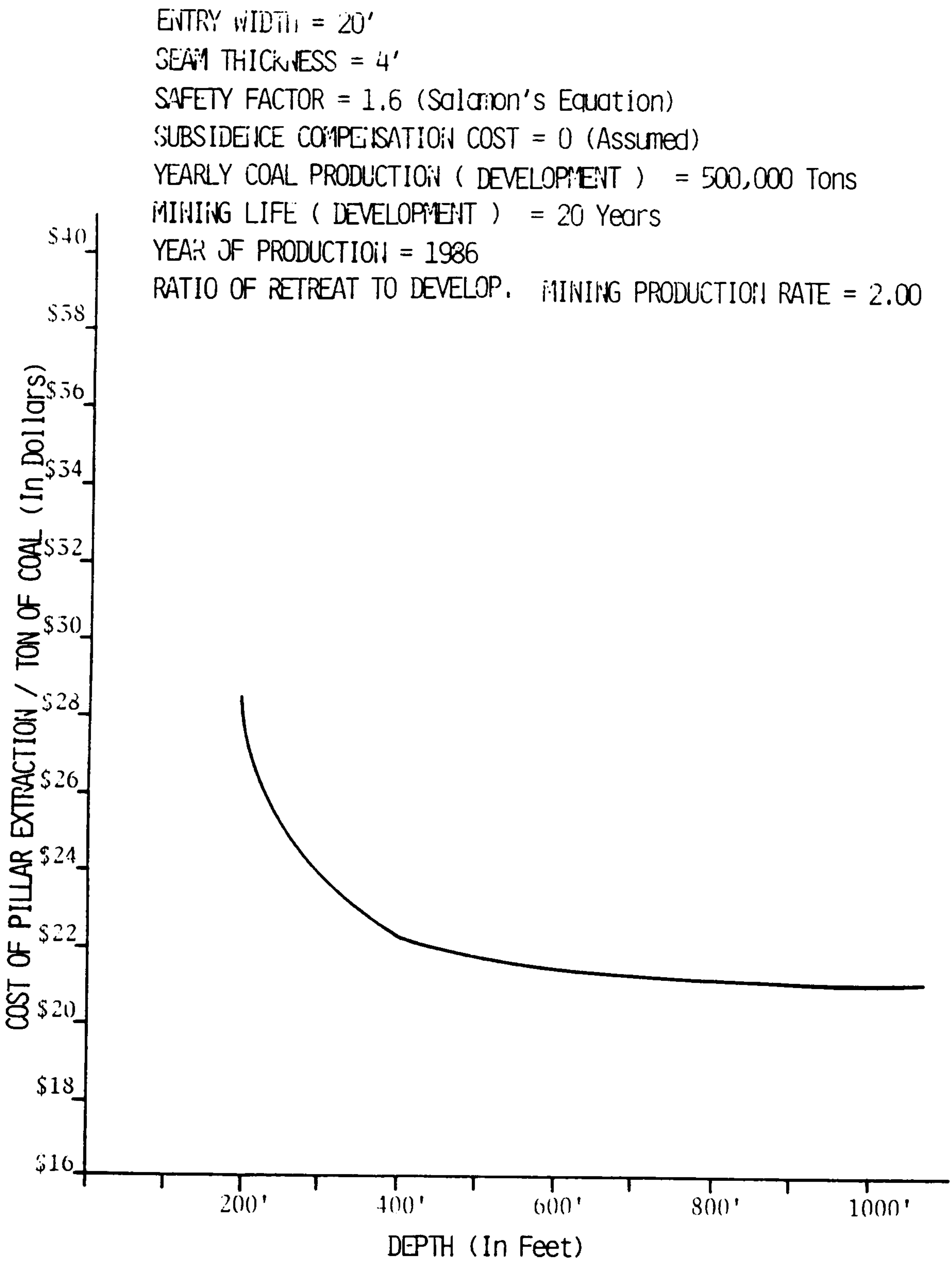
$22
$20
$18
$16
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
200'
400'
600'
800'
1000'
DEPTH (In Feet)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MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 2.00
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
$40
$38
$36
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$32
$30
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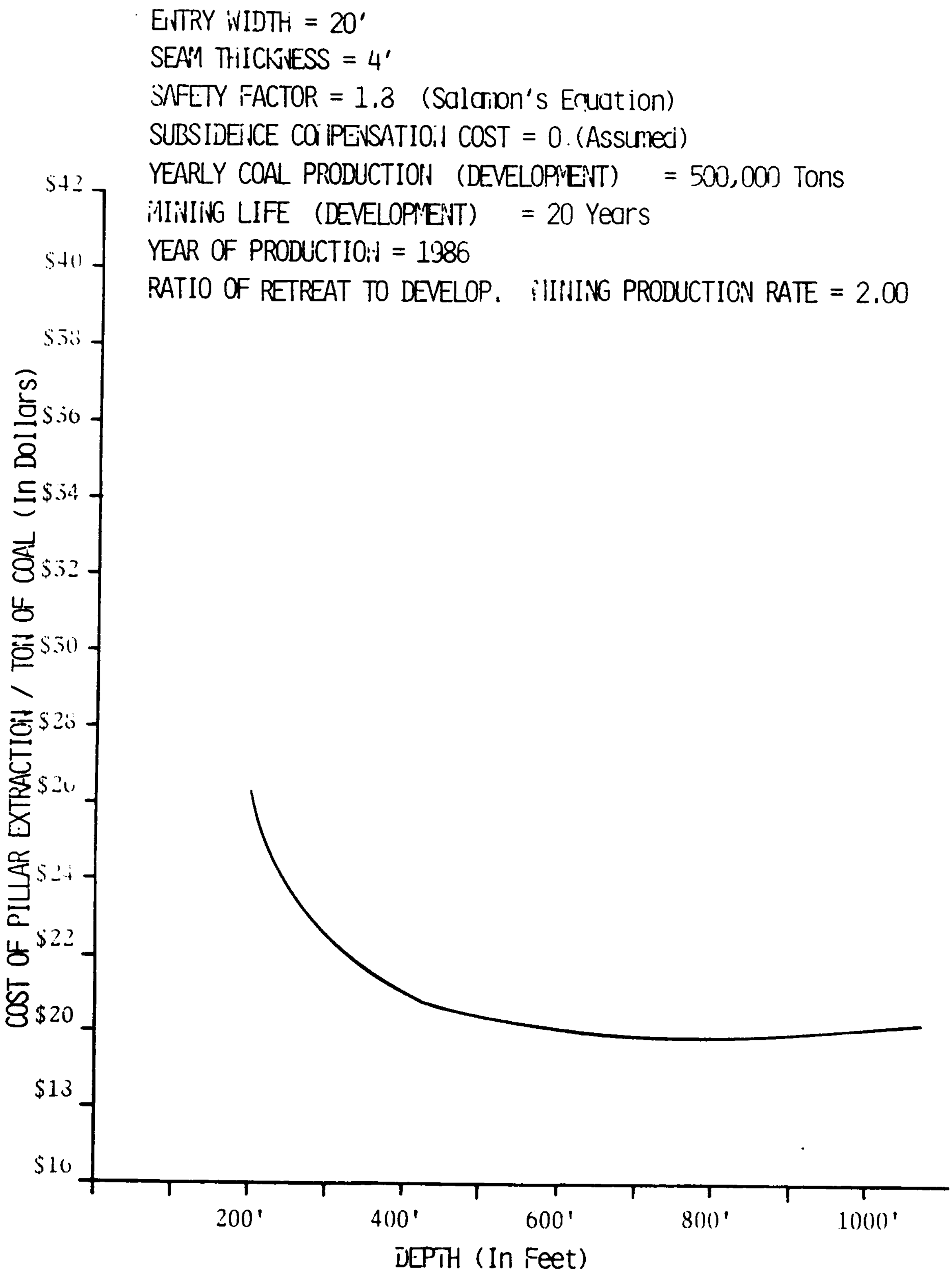
$20
$18
$16
200'
400'
600'
800'
1000'
DEPTH (In Feet)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MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 2.00
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
$42
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200'
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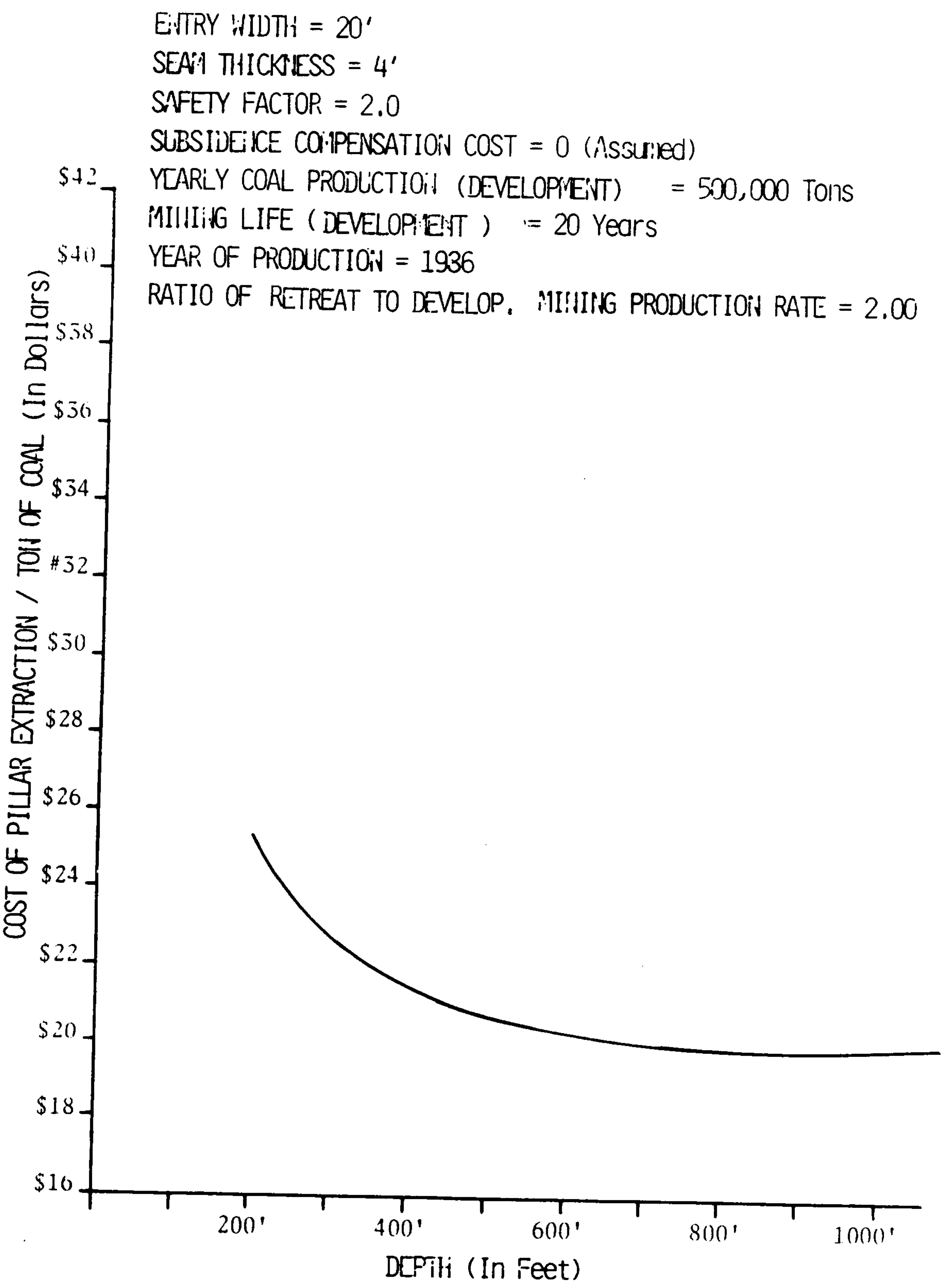
800'
1000'
DEPTH (In Feet)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SAFETY FACTOR = 2.0
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MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1936
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 2.00
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
$42
$40
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200'
400'
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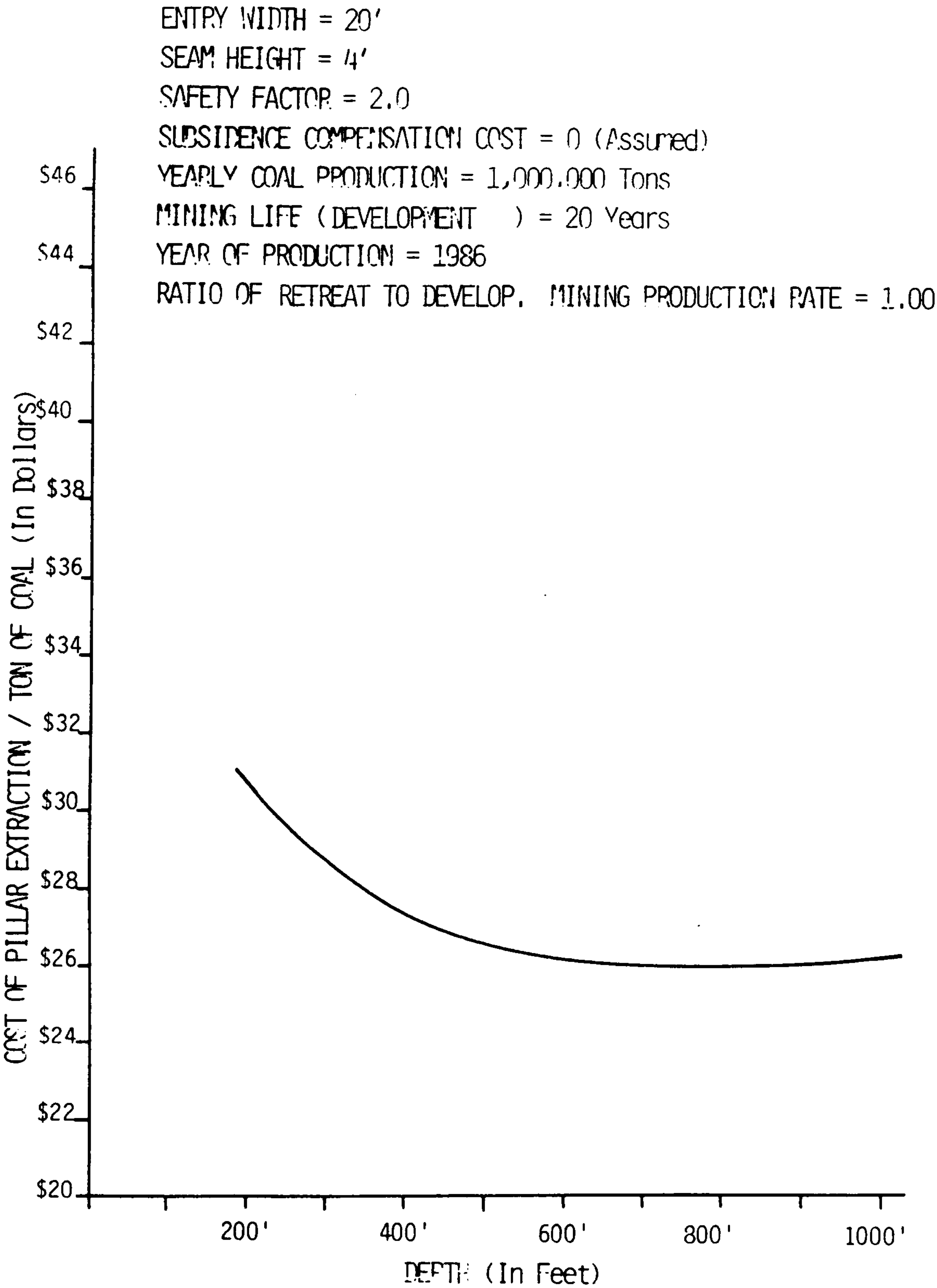
DEPTH (In Feet)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SEAM HEIGHT = 4'
SAFETY FACTOR = 2.0
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION = 1,000,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.00
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
$46
$44
$42
$40
$38
$36
$34
$32
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800'
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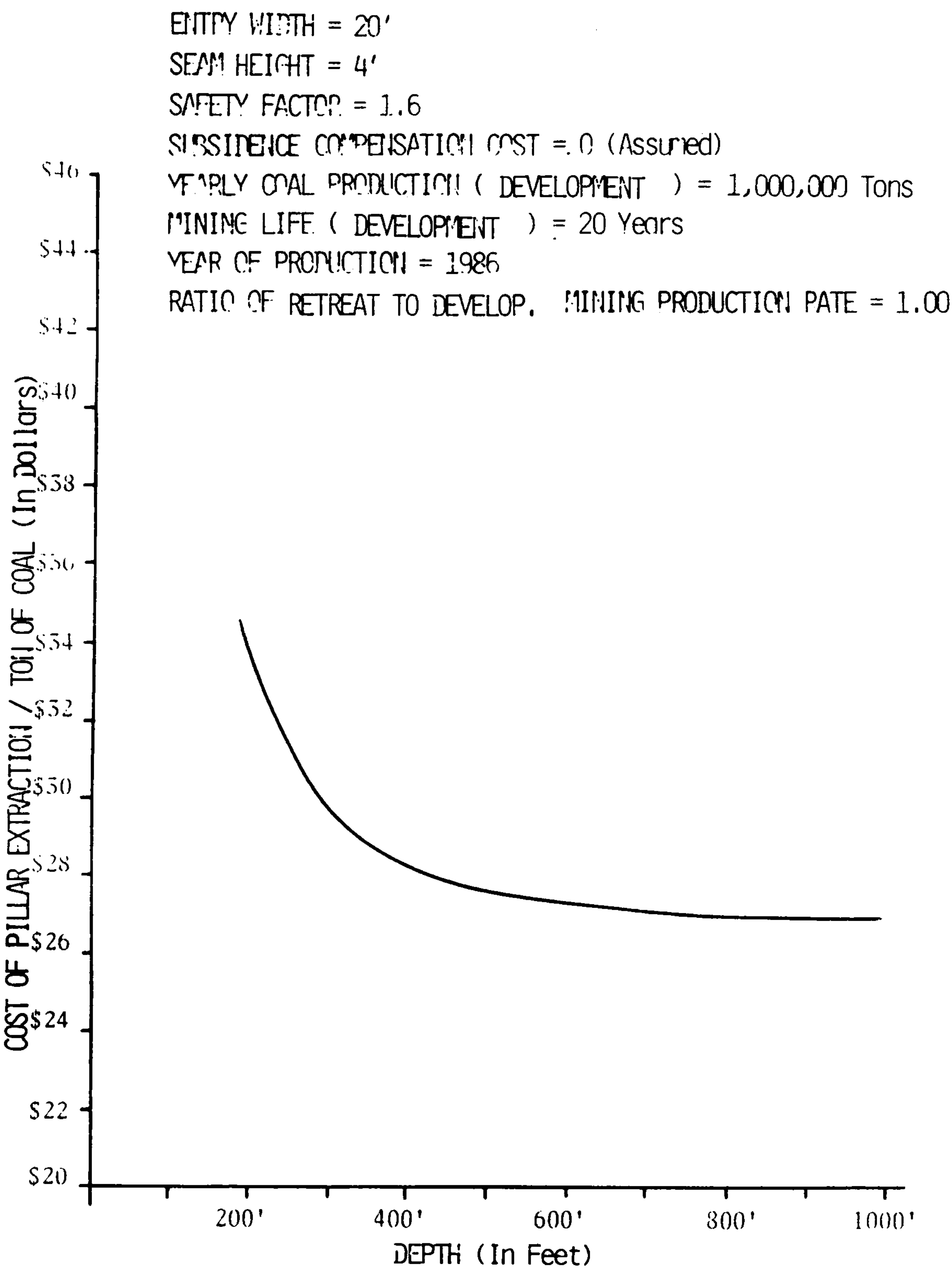
DEPTH (In Feet)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MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.00
$46
$44
$42
$40
$38
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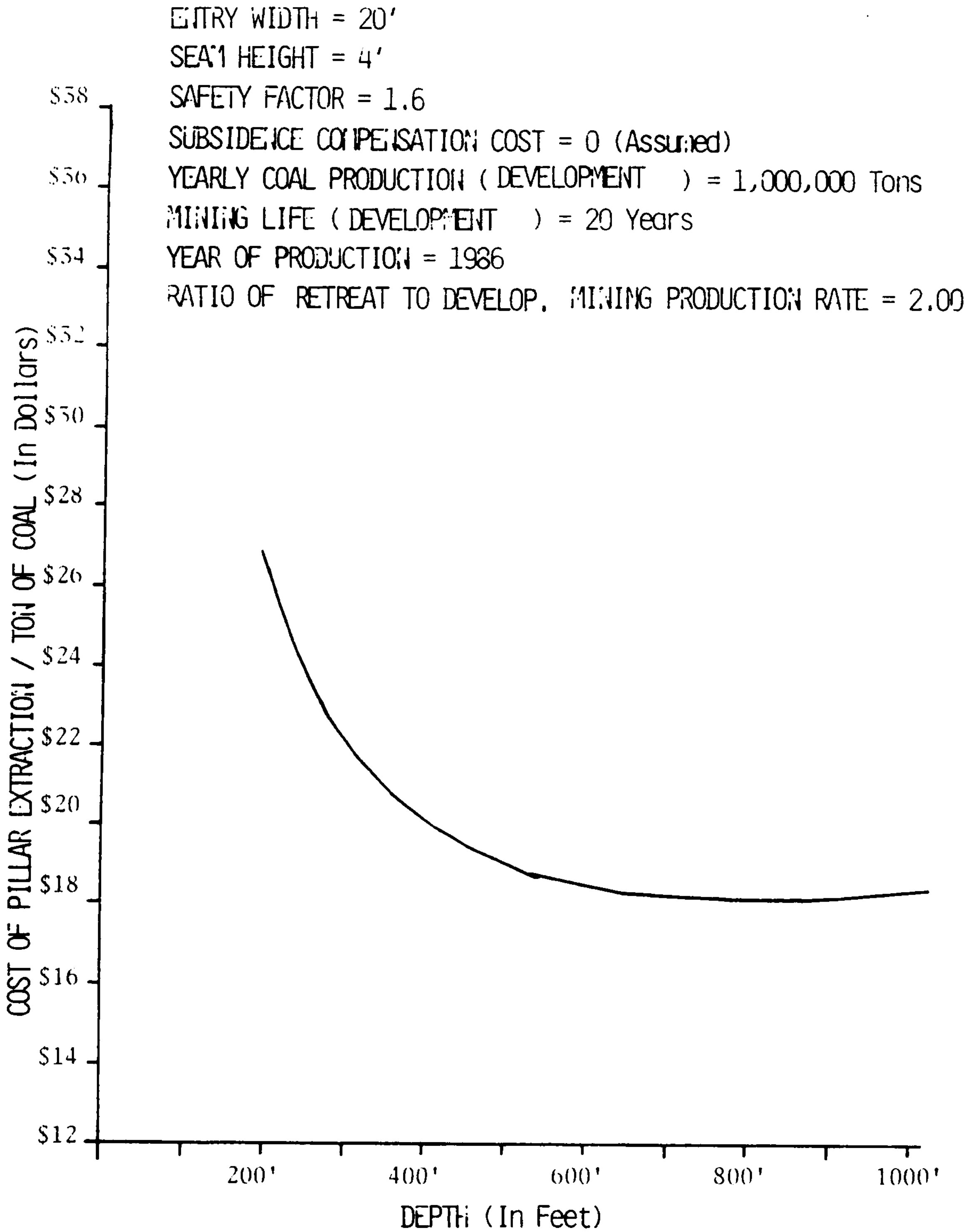
$34
$32
$30
$28
$26
$24
$22
$20
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
200'
400'
600'
800'
1000'
DEPTH (In Feet)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SAFETY FACTOR = 1.6
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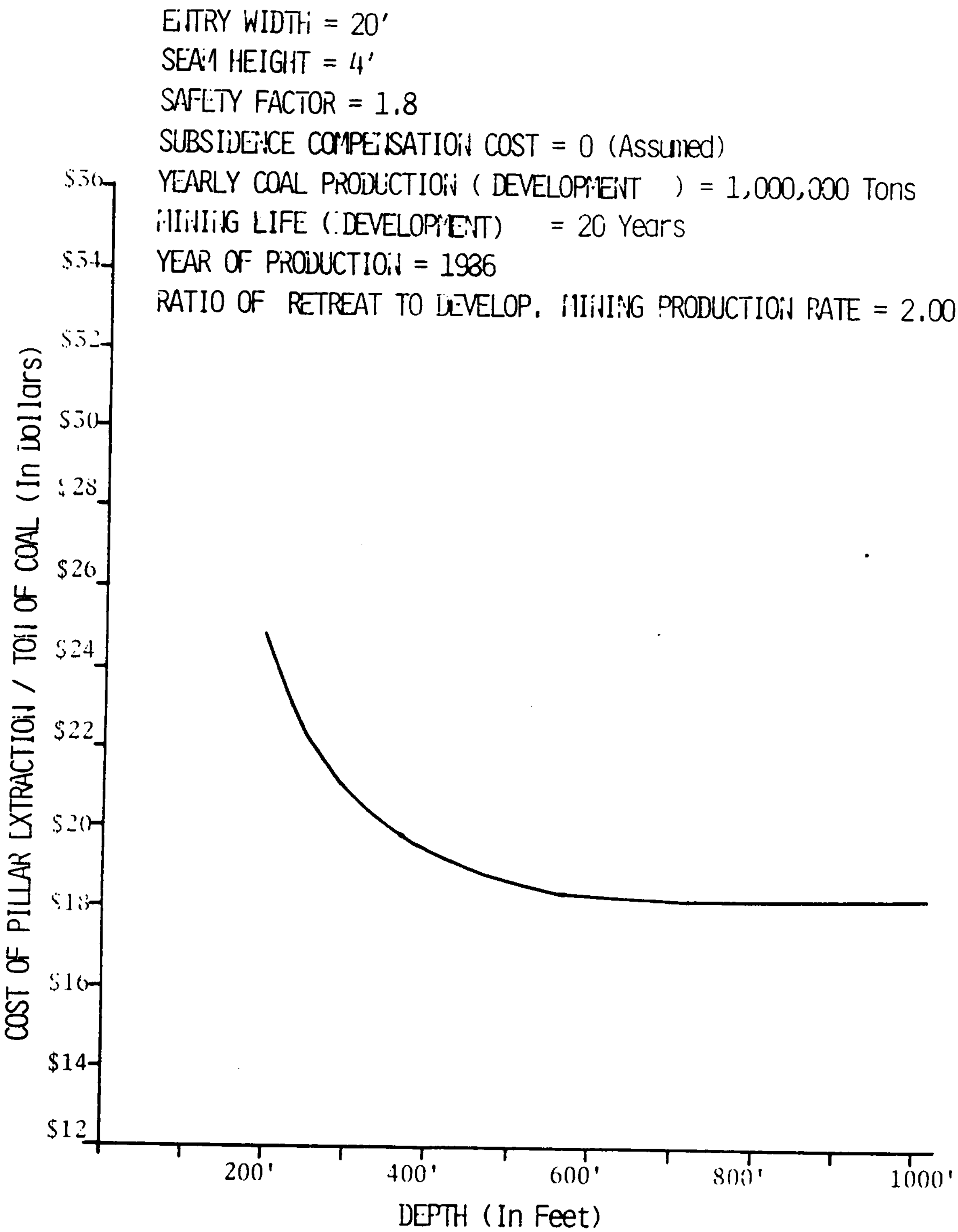
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 2.00
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
$38
$36
$34
$32
$30
$28
$26
$24
$22
$20
$18
$16
$14
$12
200'
400'
600'
800'
1000'
DEPTH (In Feet)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SAFETY FACTOR = 1.8
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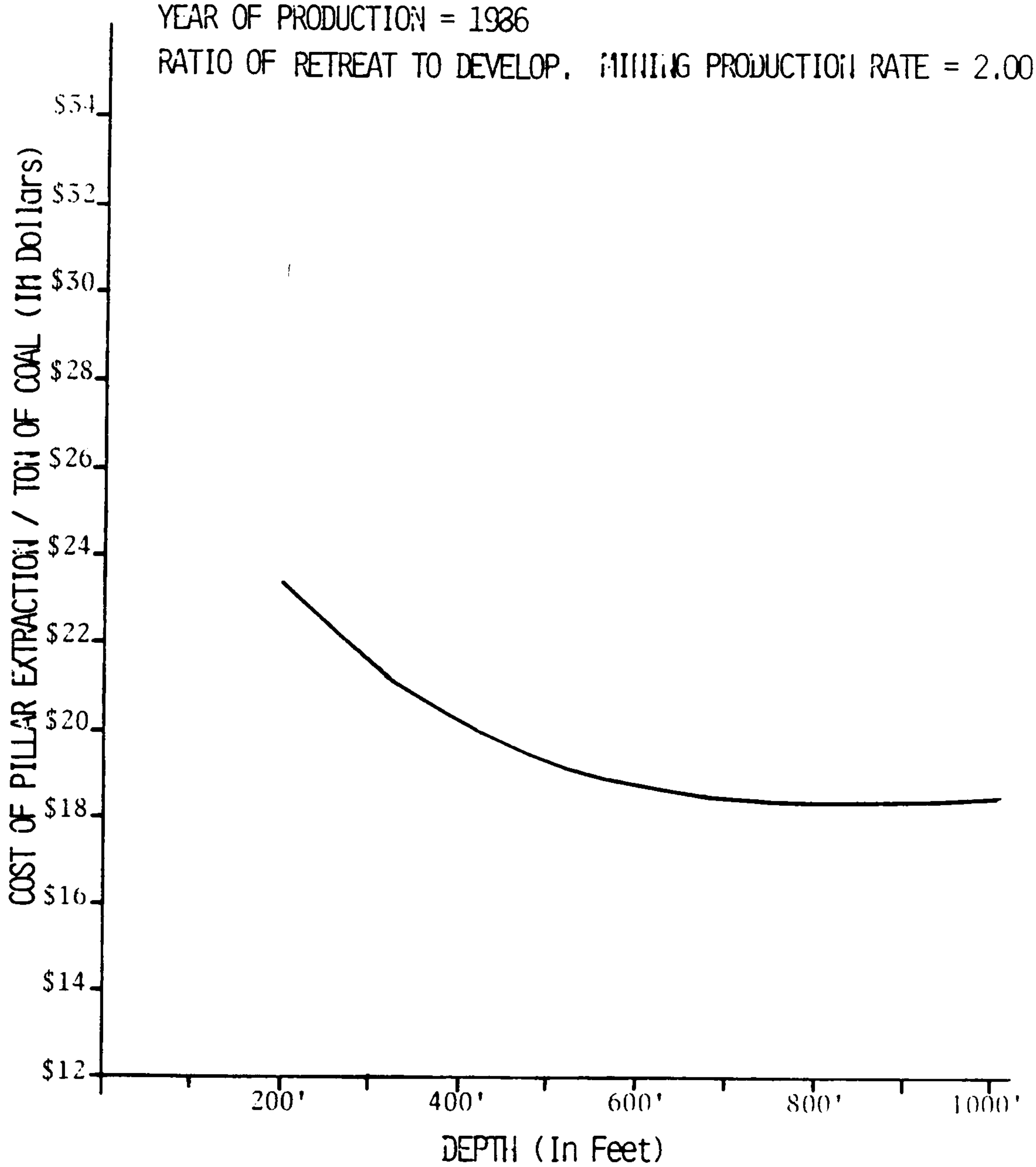
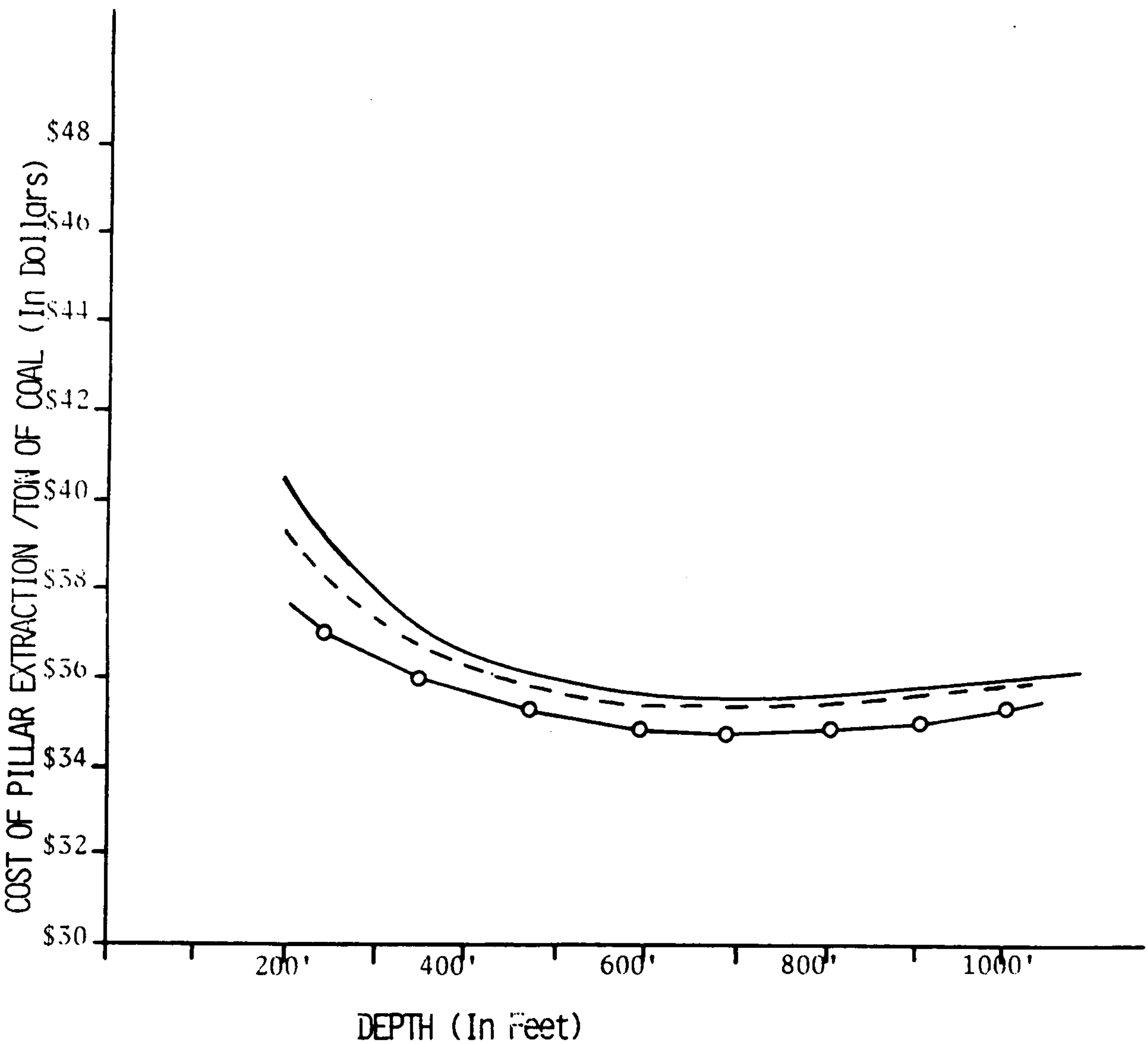
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 2.00
COST OF PILLAR EXTRACTION / TON OF COAL (In Dollars)
$36
$34
$32
$30
$28
$26
$24
$22
$20
$18
$16
$14
$12
200'
400'
600'
800'
1000'
DEPTH (In Feet)

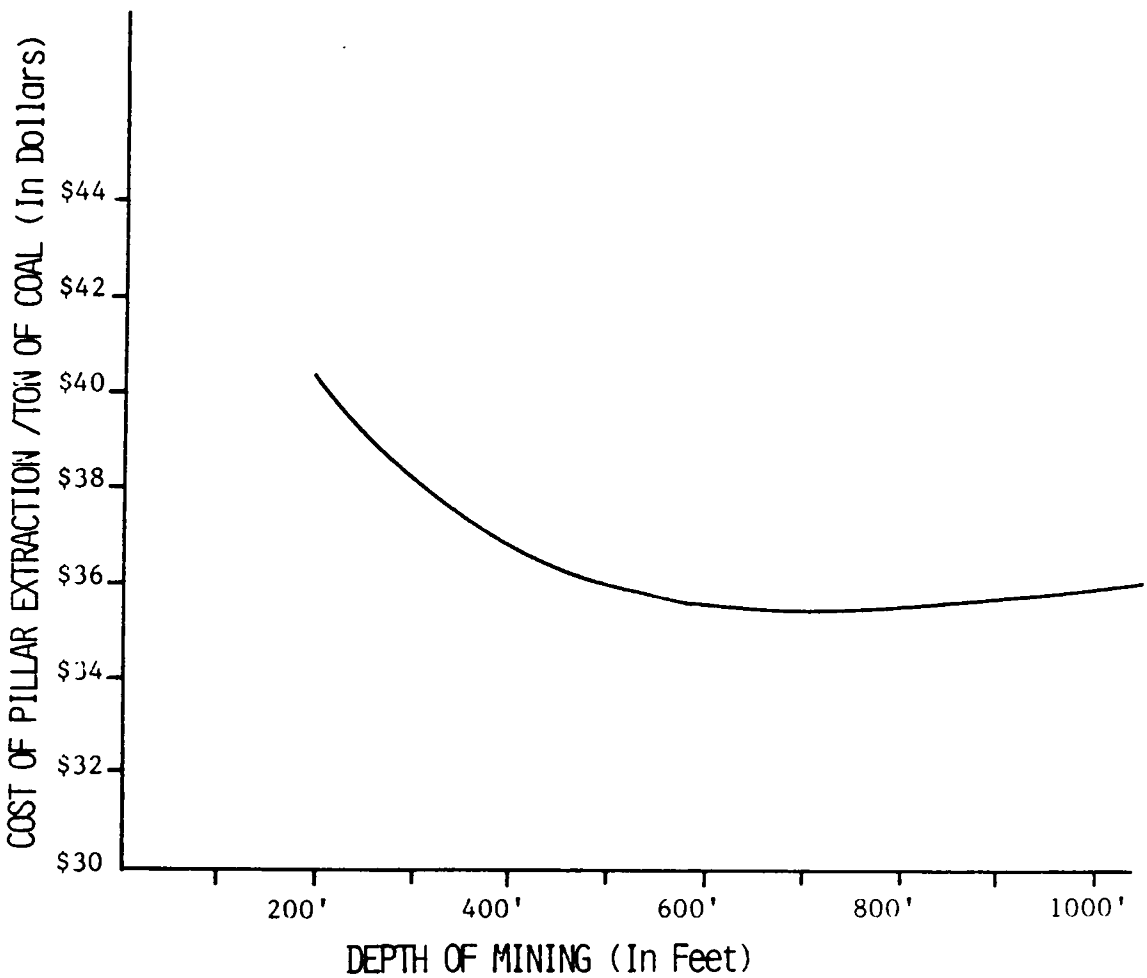
ENTRY WIDTH = 20'
SEAM THICKNESS = 4'
SAFETY FACTOR = 2.0
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 1,000,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 2.00



ENTRY WIDTH = 20'
SEAM THICKNESS = 5'
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION = 150,000 Tons (DEVELOPMENT)
MINING LIFE (DEVELOPMENT) = 20 Years
RATIO OF RETREAT TO DEVELOP. MINING RODUCTION RATES = 1.00
SAFETY FACTOR = 1.6
= 1.8
= 2.0



ENTRY WIDTH = 20'
SEAM THICKNESS = 5'
SAFETY FACTOR = 1.6 (Salamon's Equation)
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 150,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1985
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.00



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SUBSIDENCE COMPENSATION COST = 0 (Assumed)
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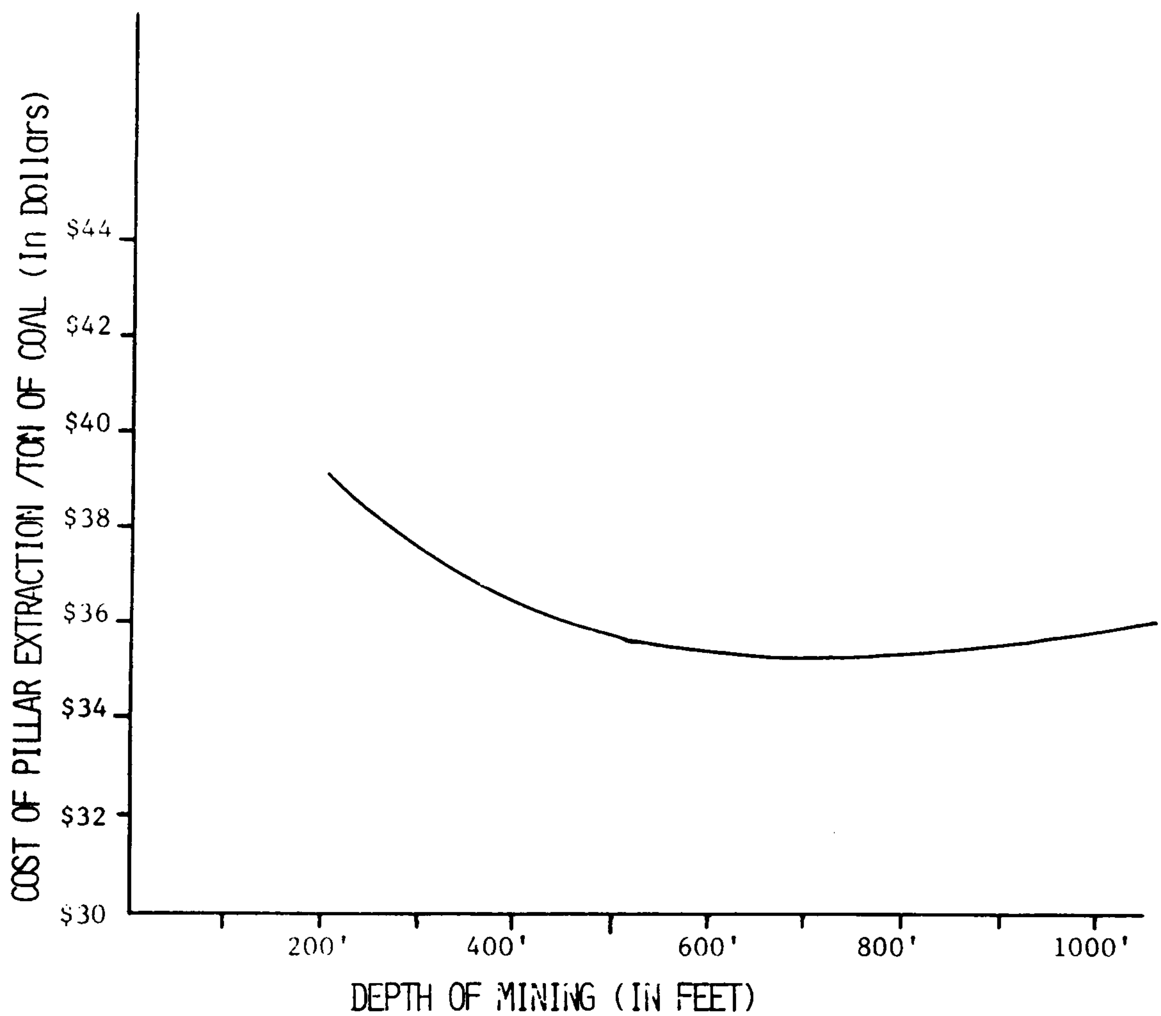
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.00



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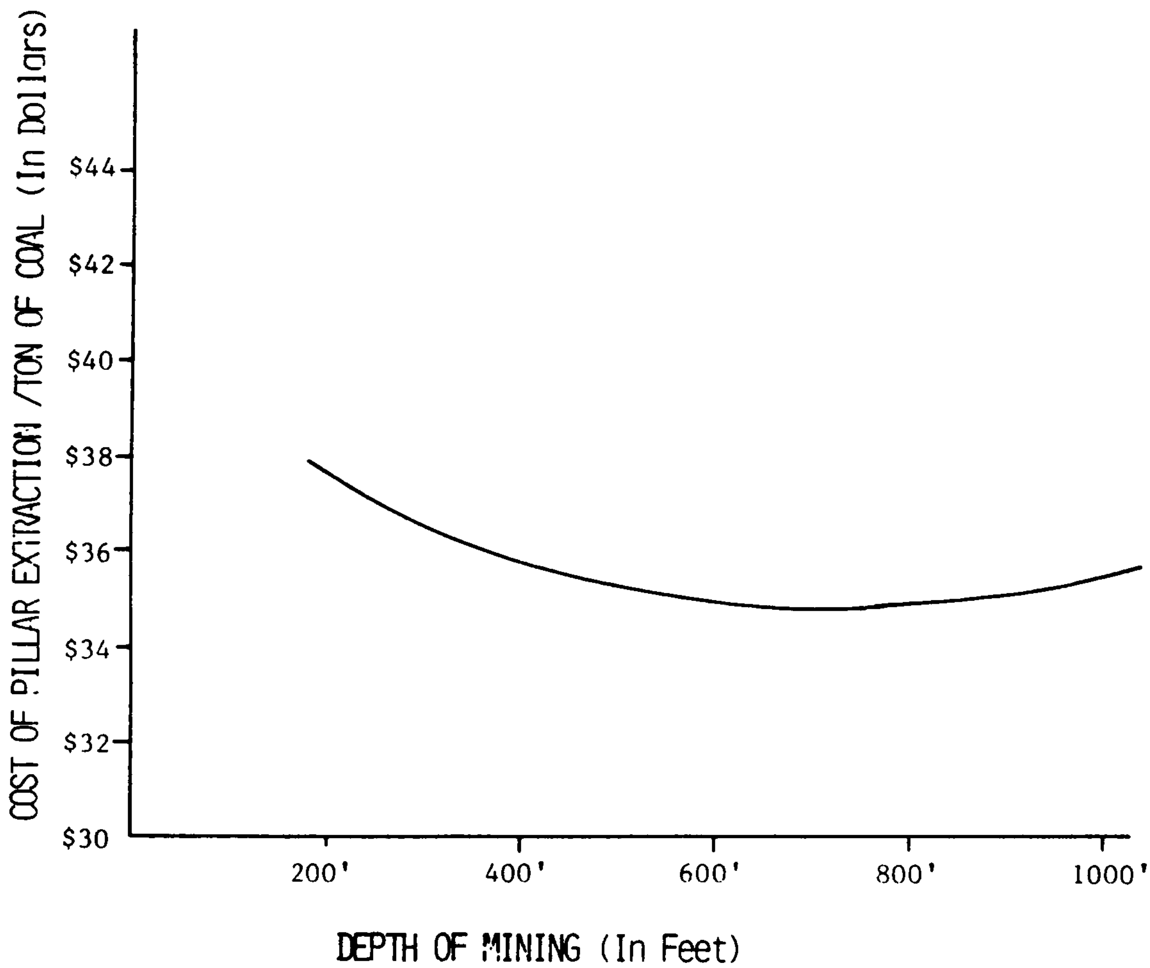
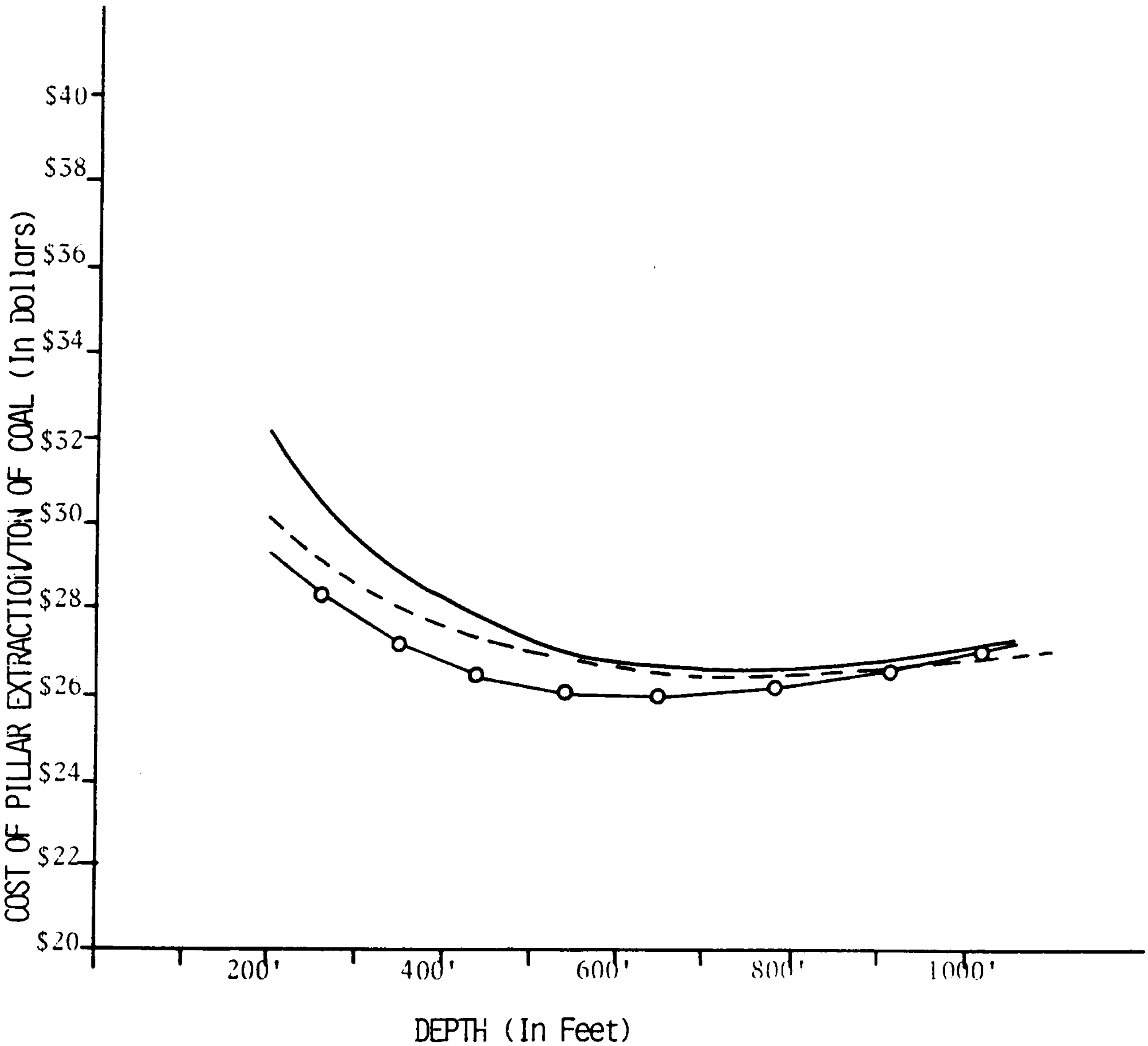
SEAM THICKNESS = 5'
SAFETY FACTOR = 2.0 (Salamon's Equation)
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 150,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.00



ENTRY WIDTH = 20′
SEAM THICKNESS = 5′
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION = 150,000 Tons
MINING LIFE (DEVELOPMENT) = 20 Years
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.50
SAFETY FACTOR = 1.6
= 1.8
= 2.0



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SAFETY FACTOR = 1.6 (Salamon's Equation)

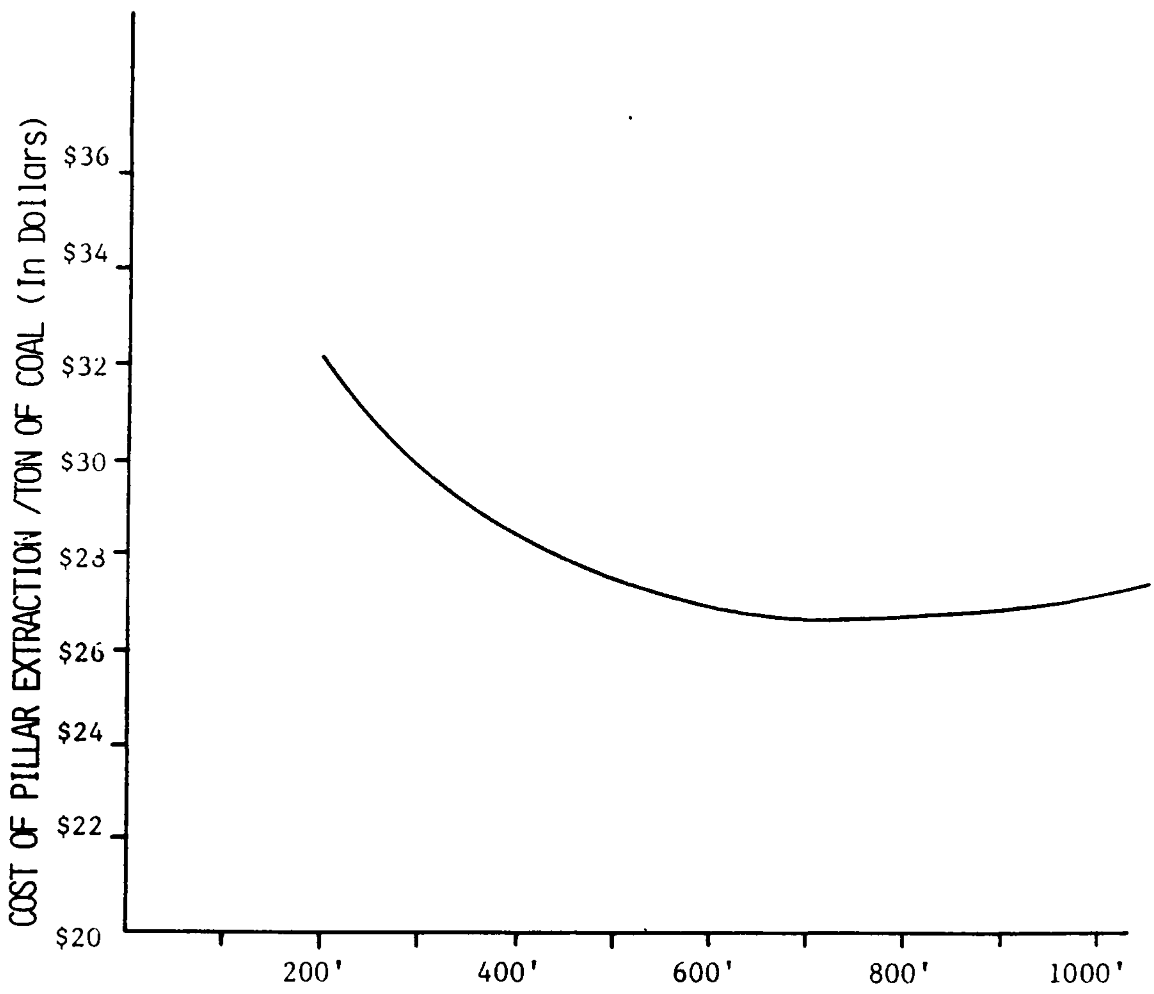
SUBSIDENCE COMPENSATION COST = 0 (Assumed)

YEARLY COAL PRODUCTION (DEVELOPMENT) = 150,000 Tons

MINING LIFE (DEVELOPMENT) = 20 Years

YEAR OF PRODUCTION = 1986

RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.5



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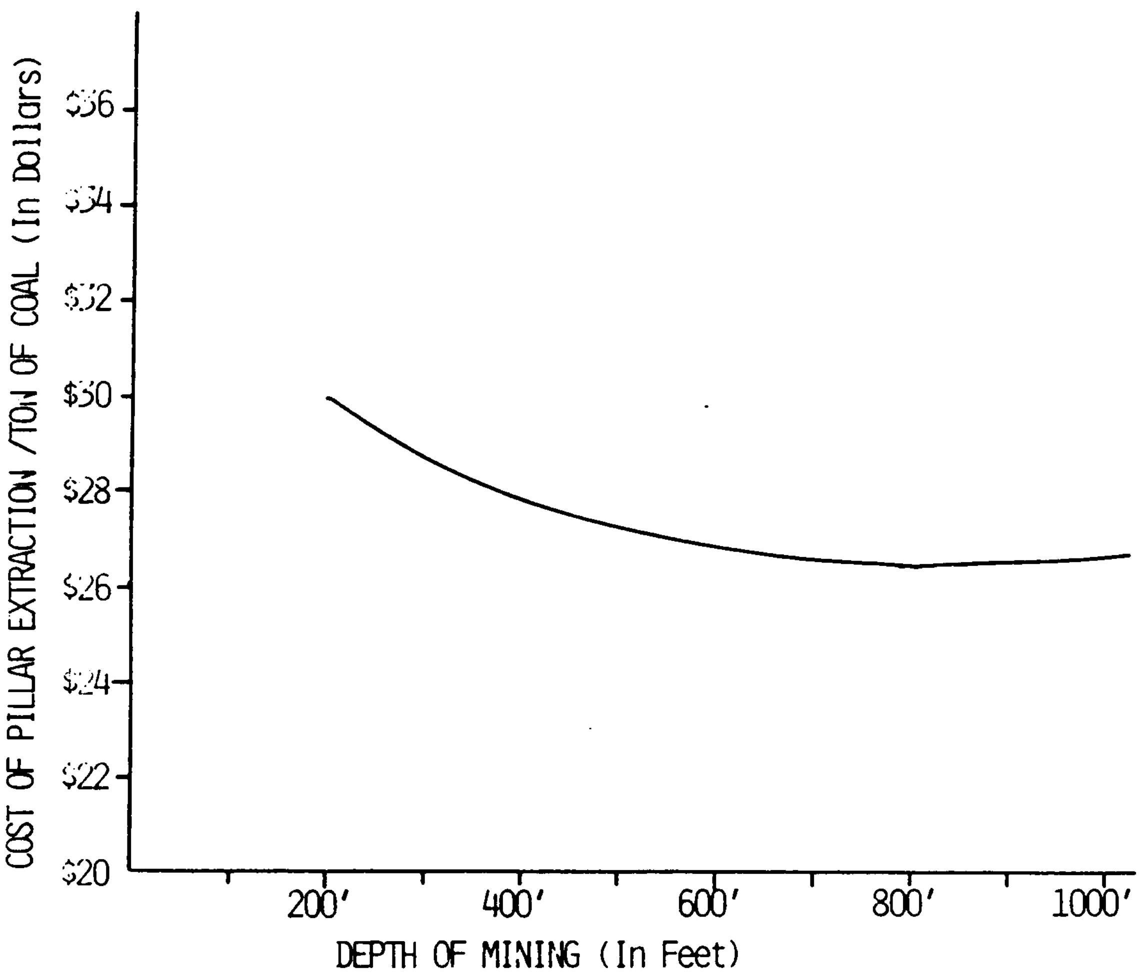
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.5



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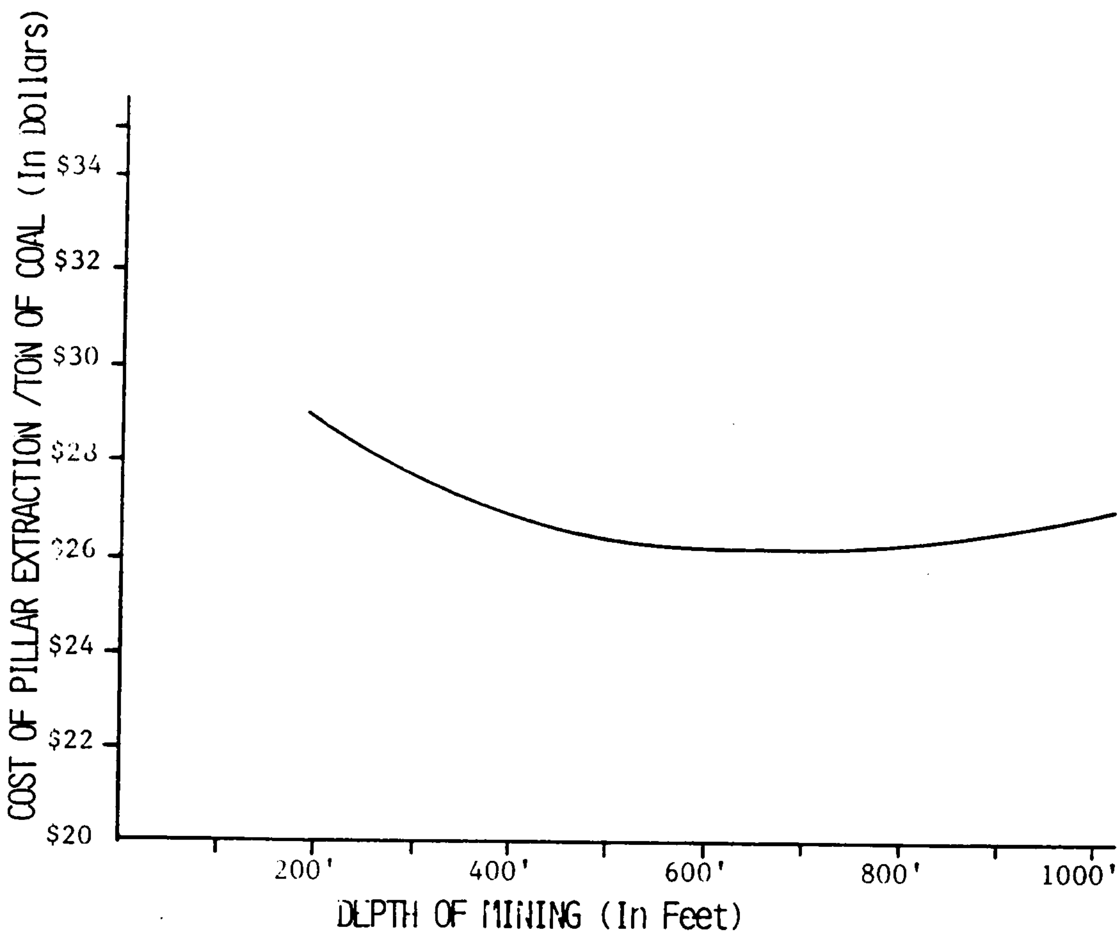
MINING LIFE (DEVELOPMENT) 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.5



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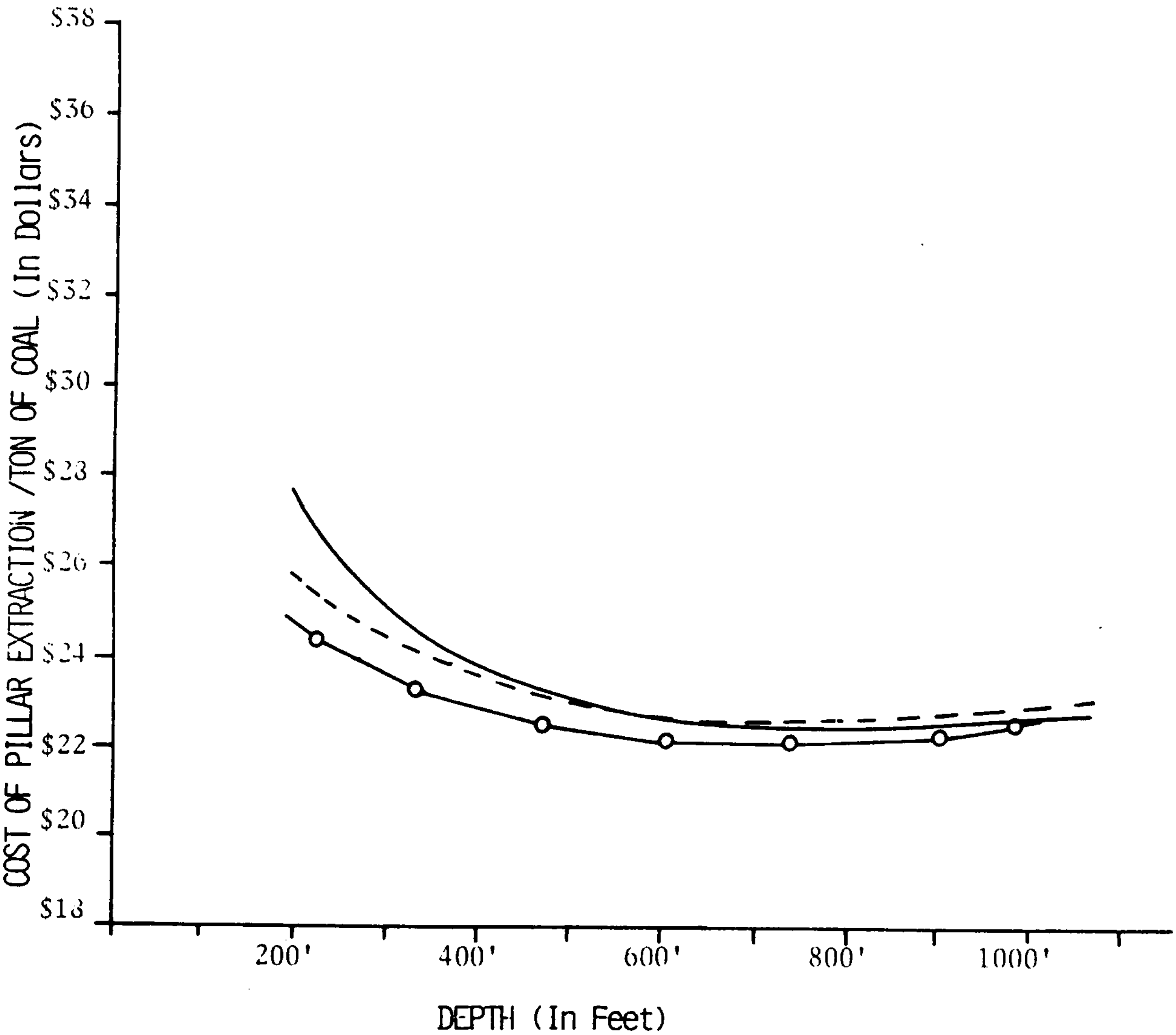
SAFETY FACTOR = 1.6
= 1.8
= 2.0



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YEAR OF PRODUCTION = 1986
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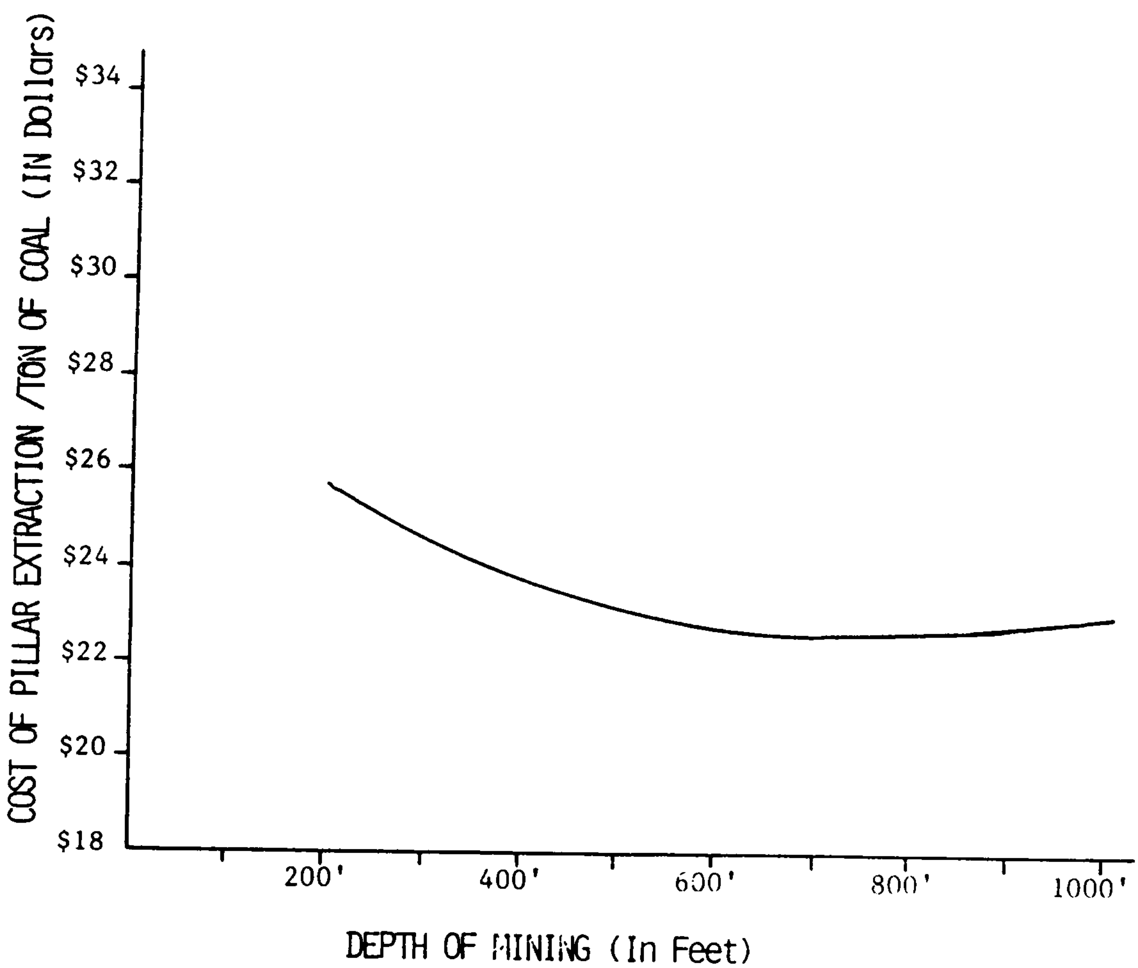

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SUBSIDENCE COMPENSATION COST = 0 (Assumed)

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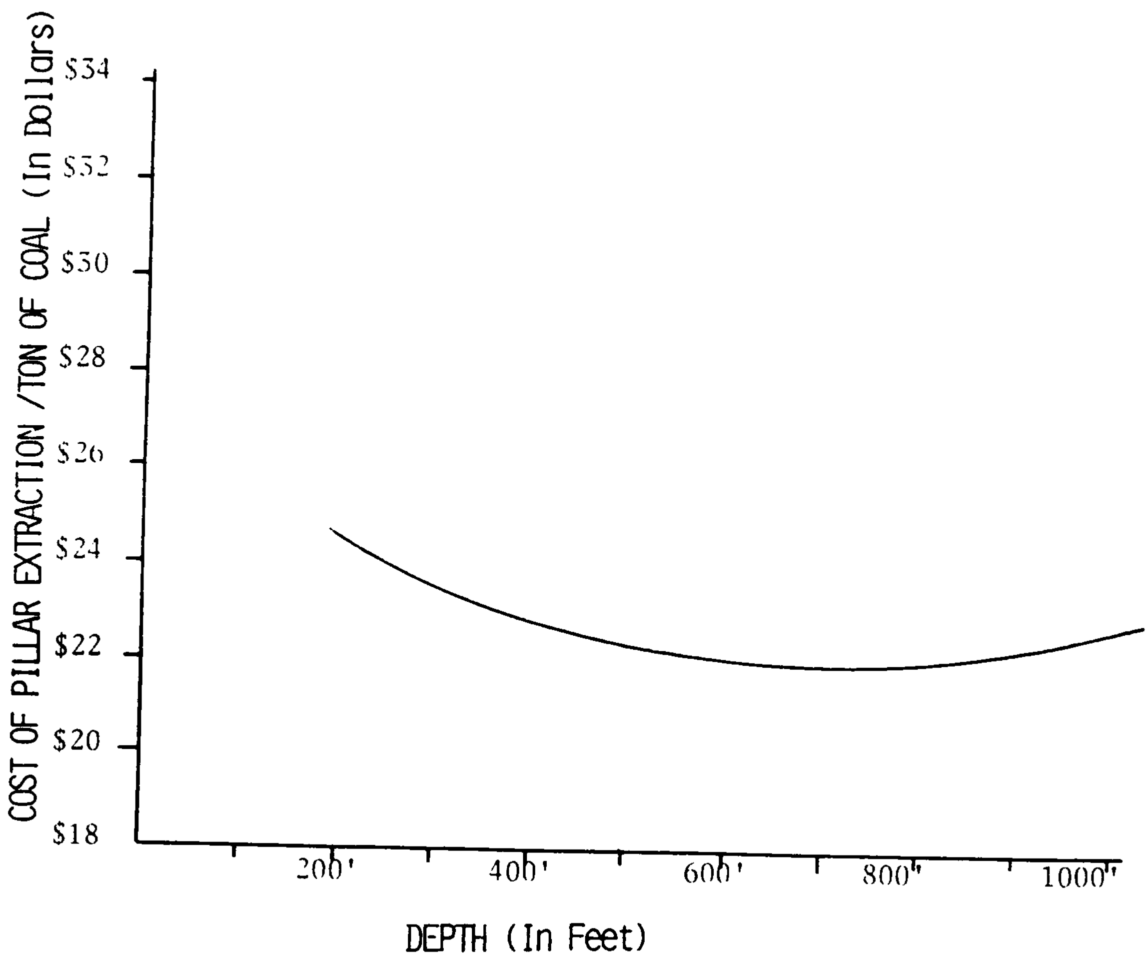
MINING LIFE (DEVELOPMENT) = 20 Years

YEAR OF PRODUCTION = 1986

RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 2.00



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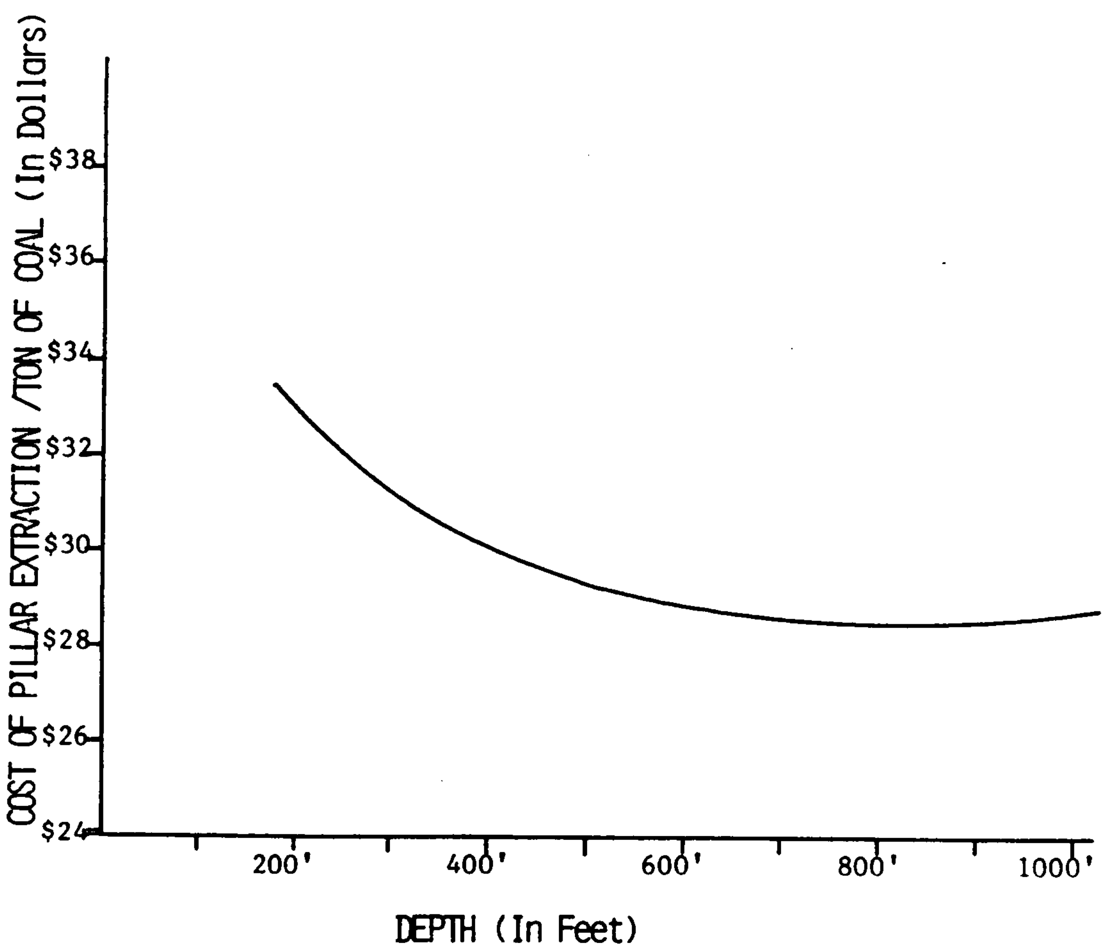
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.00



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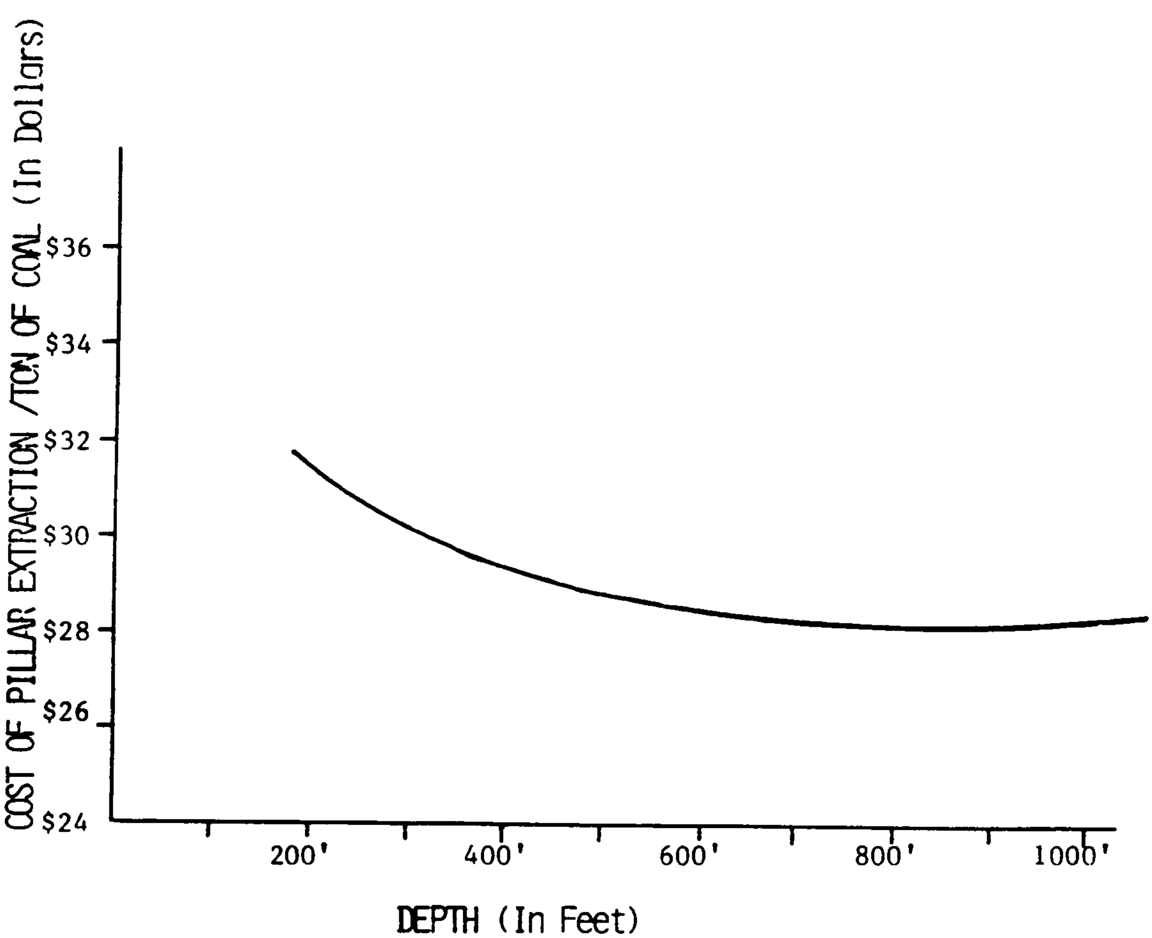
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION =1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.00



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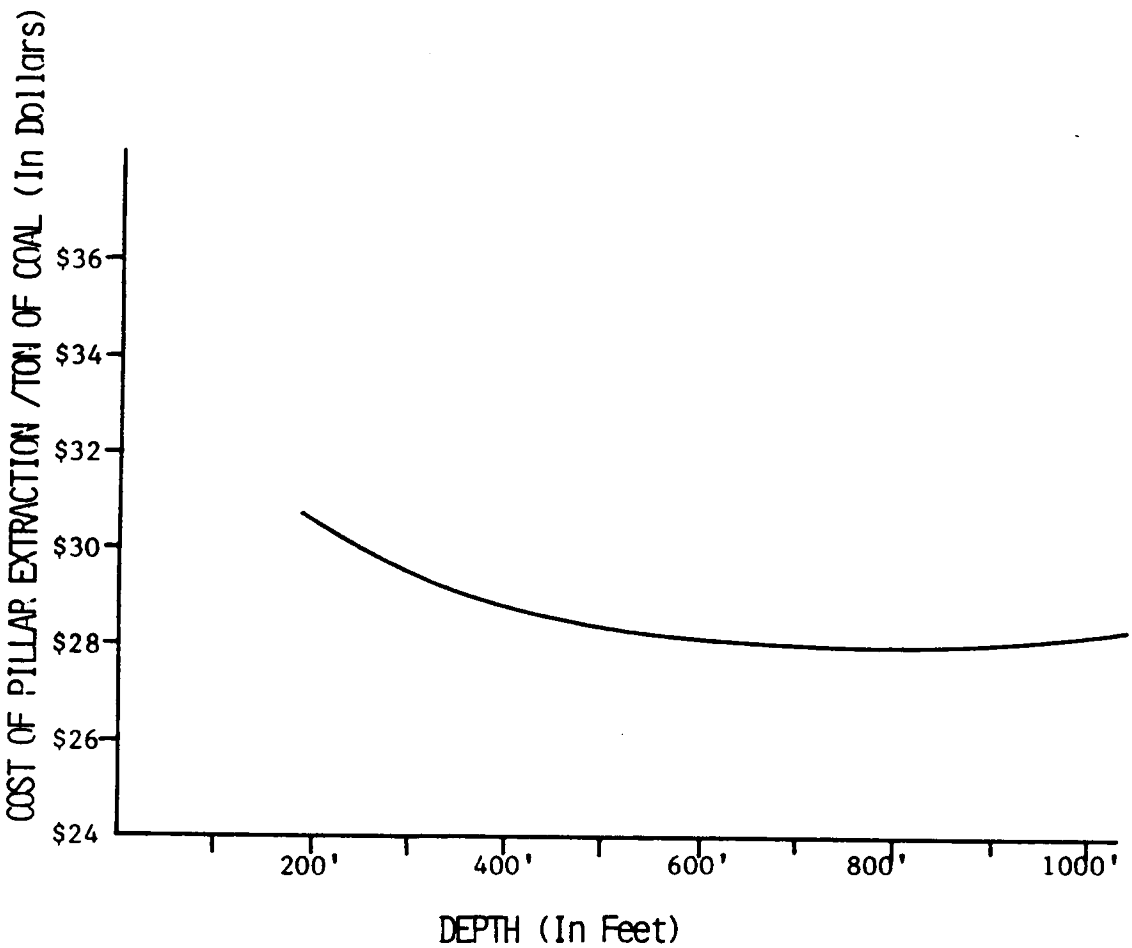
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.00



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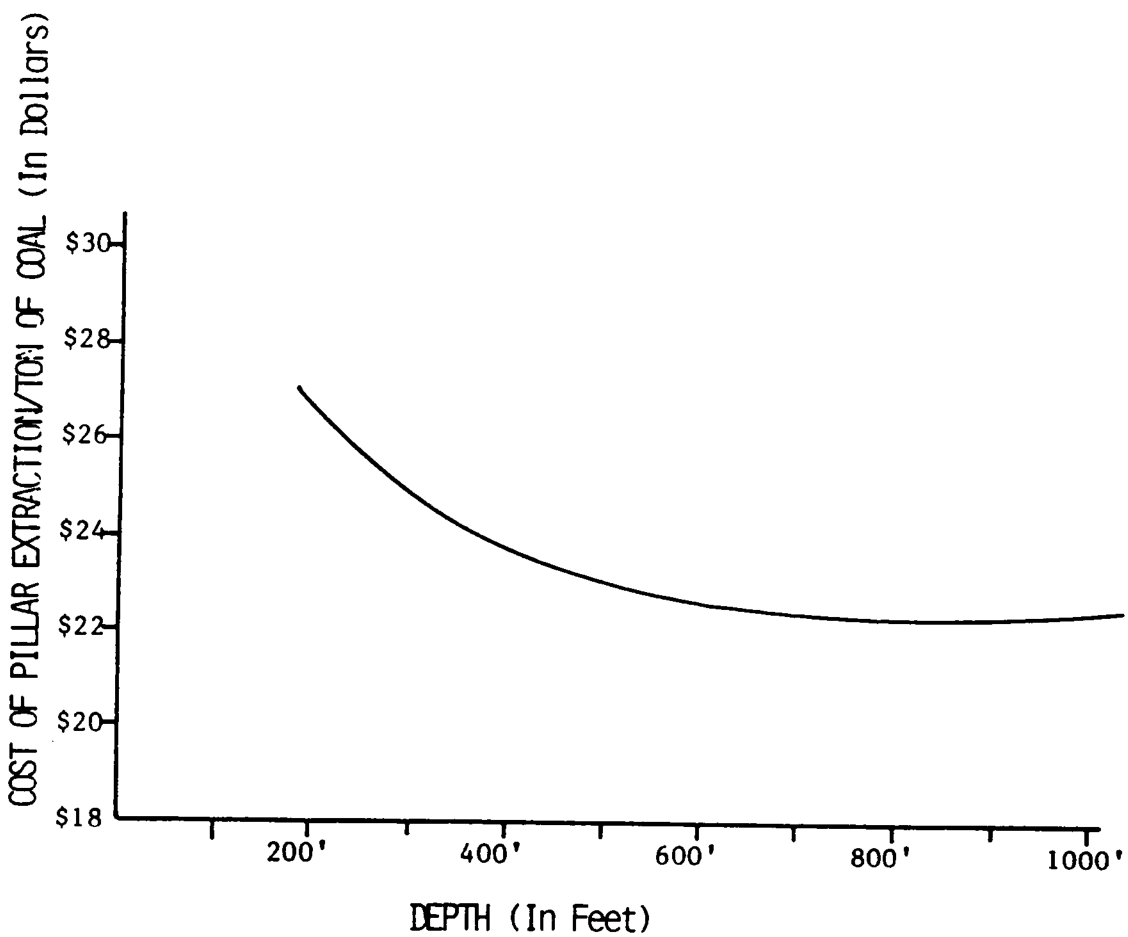
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.5



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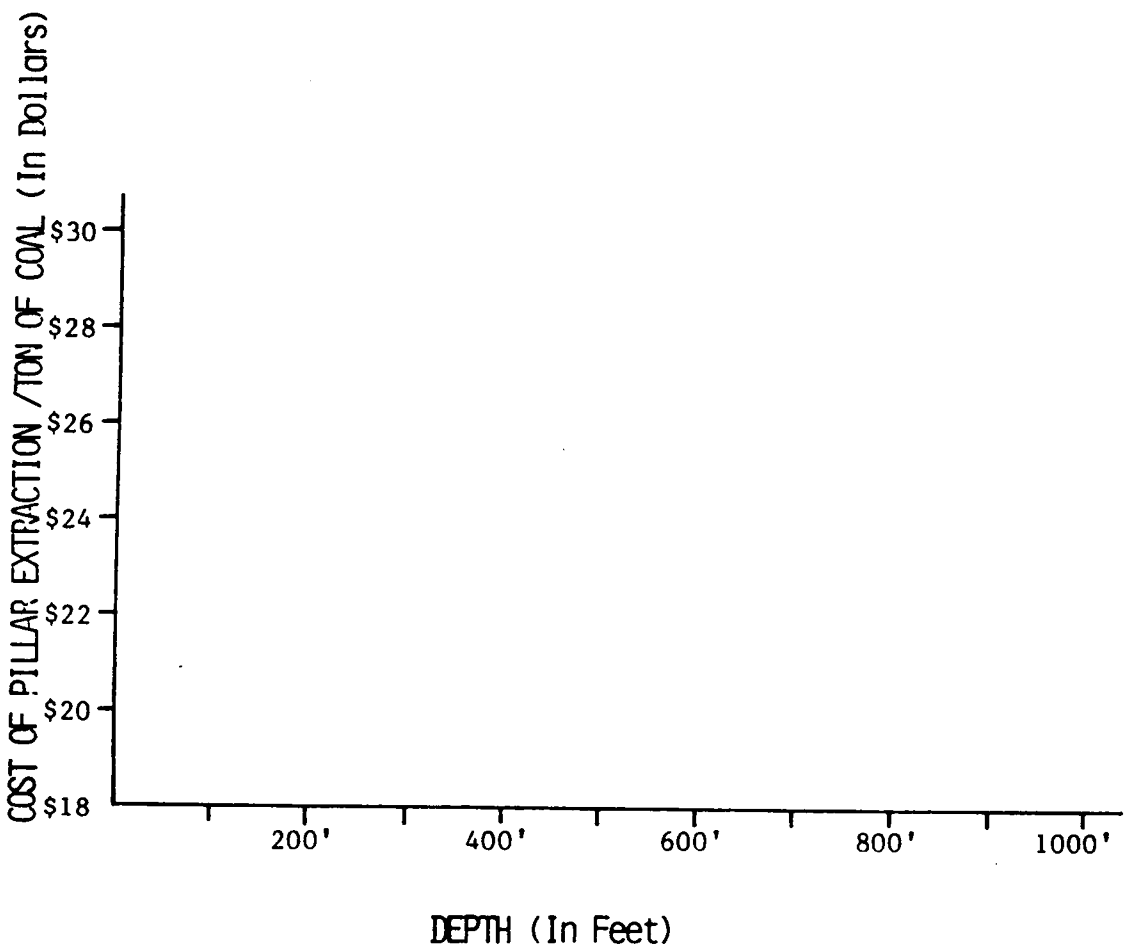
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.5



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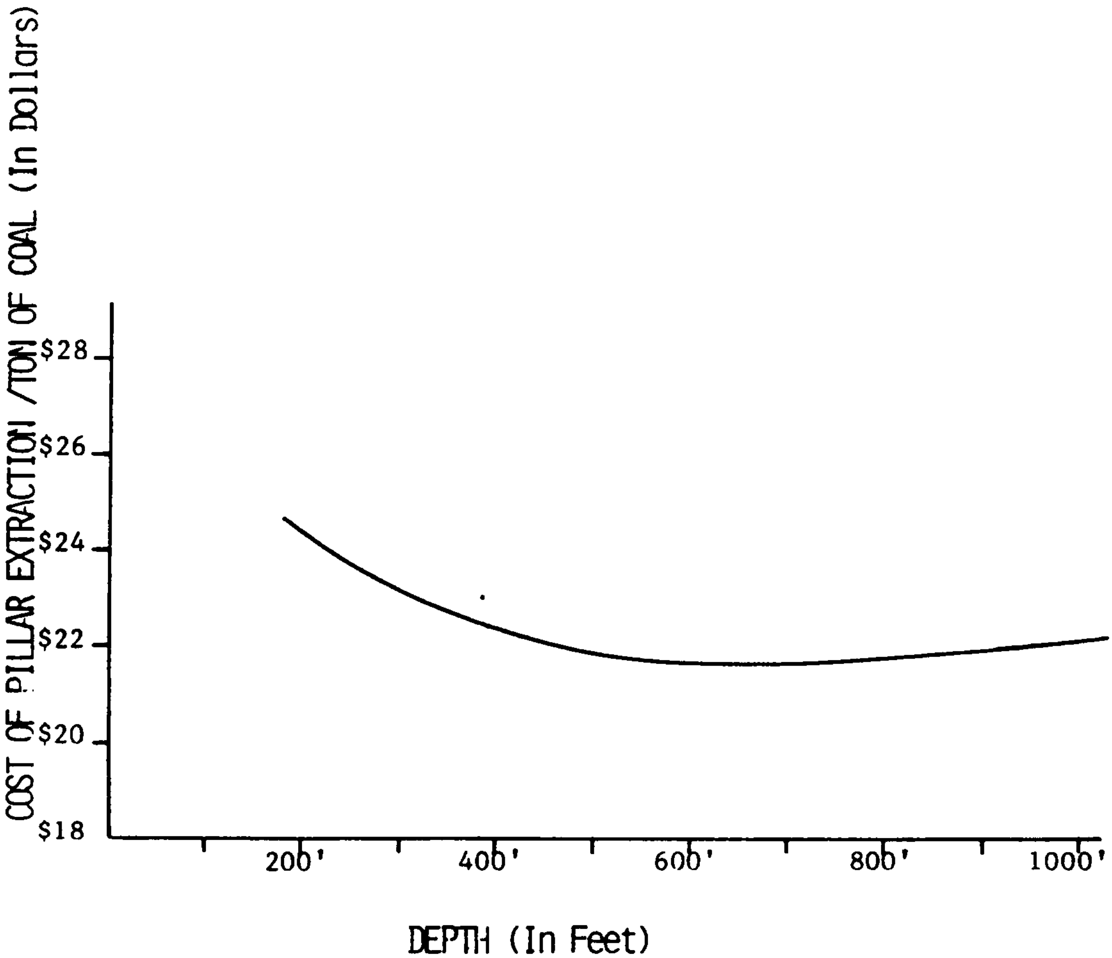
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.50



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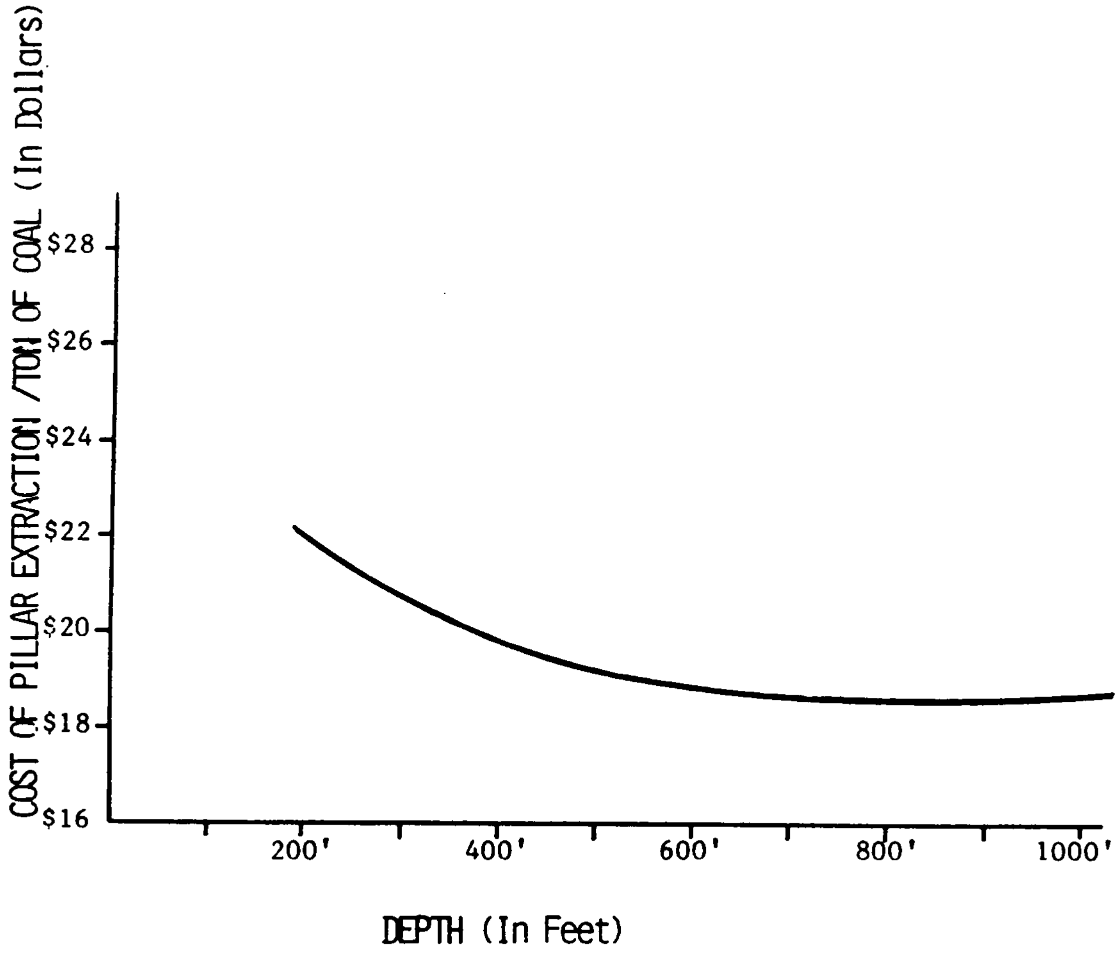
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 2.00



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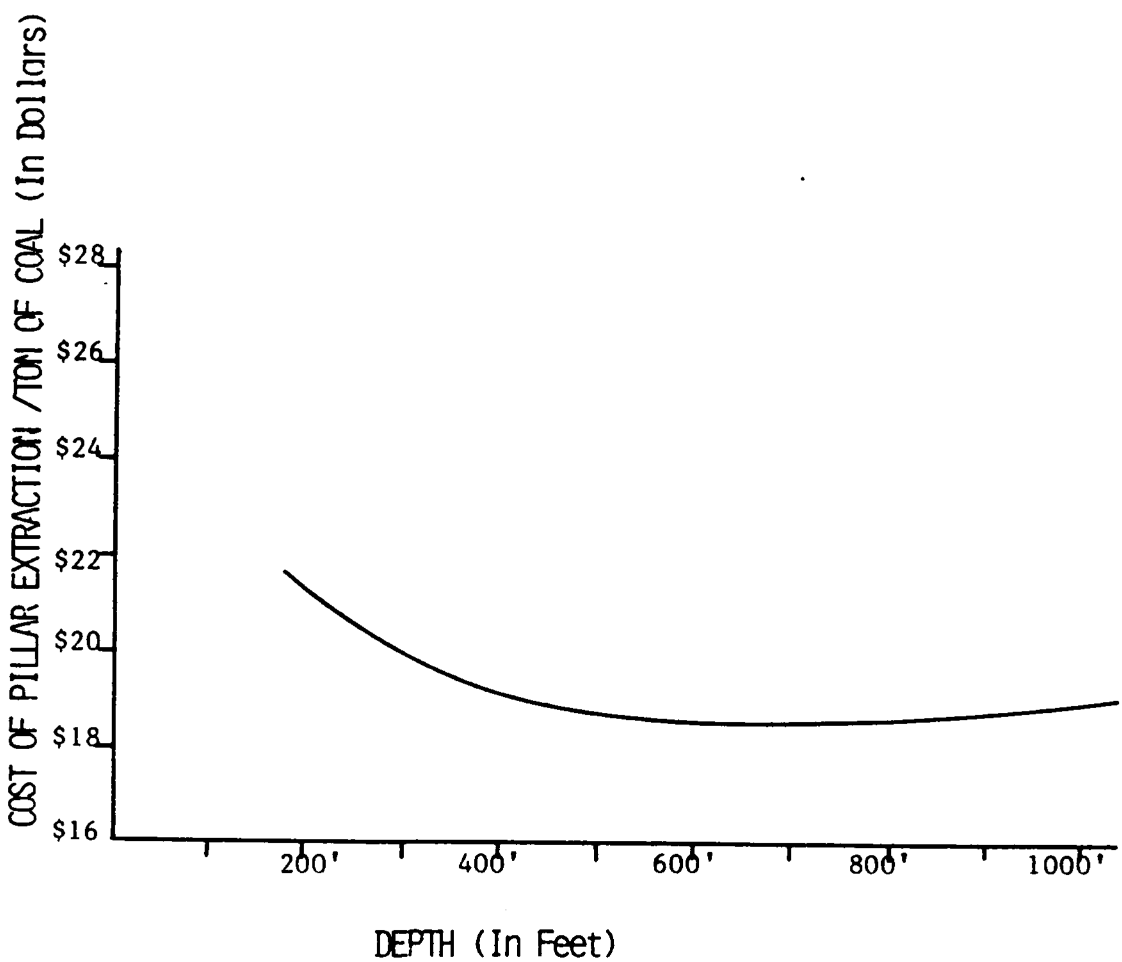
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 2.00



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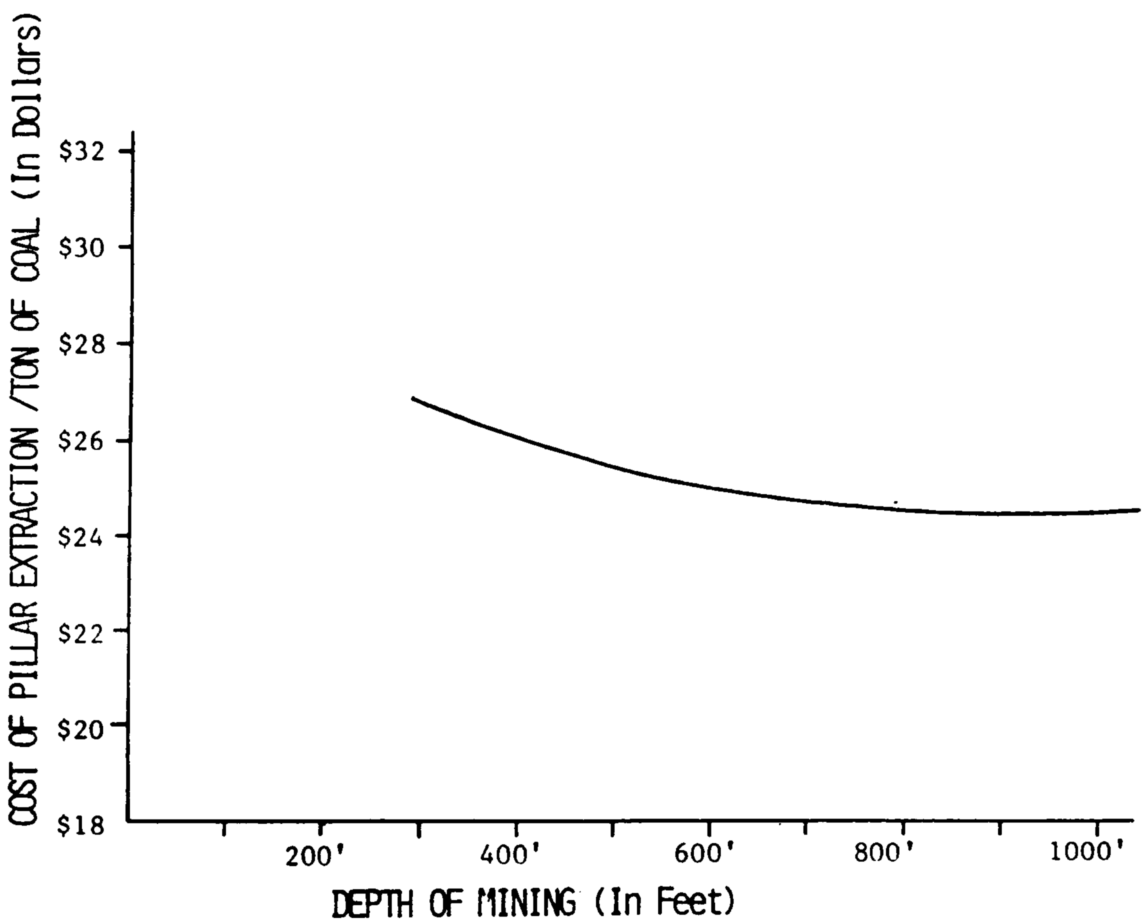
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 1 Million Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.0



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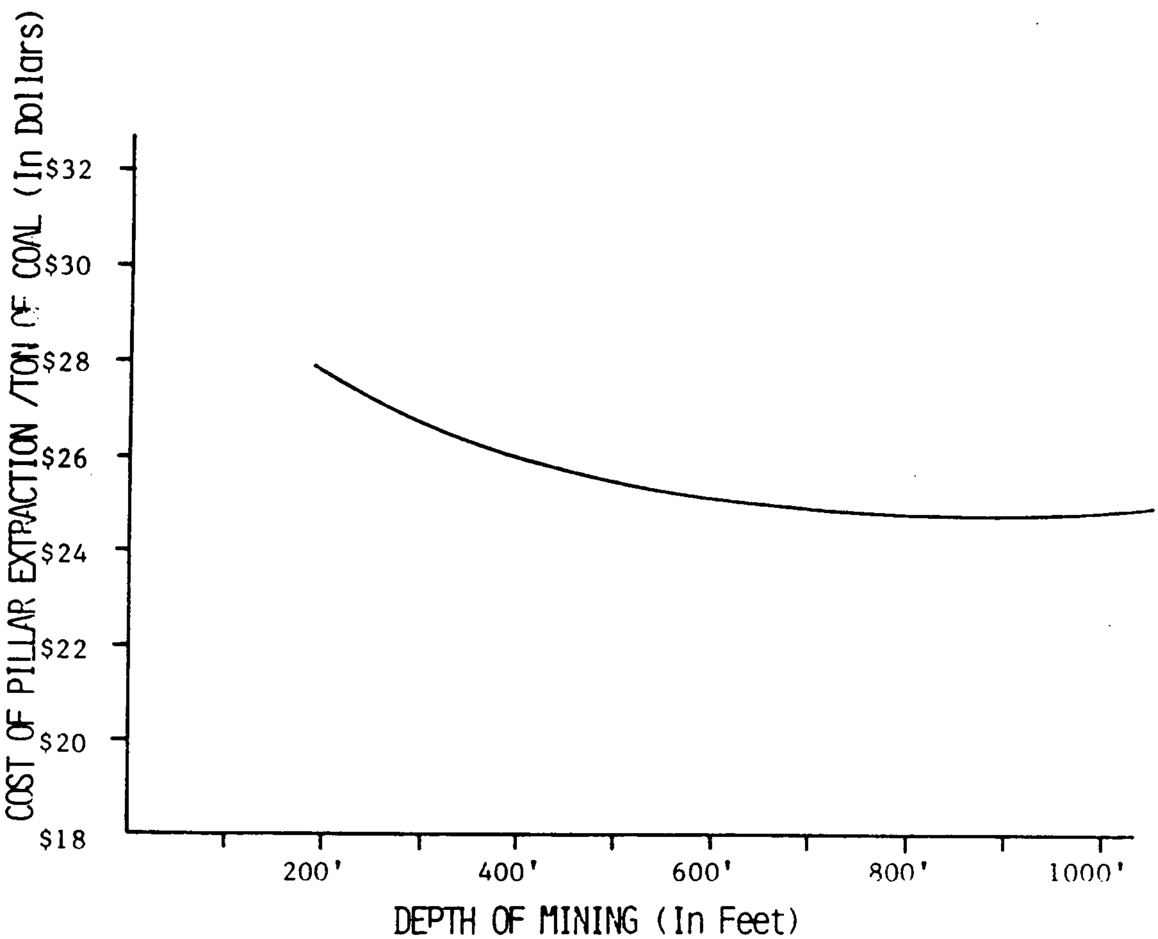
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.0



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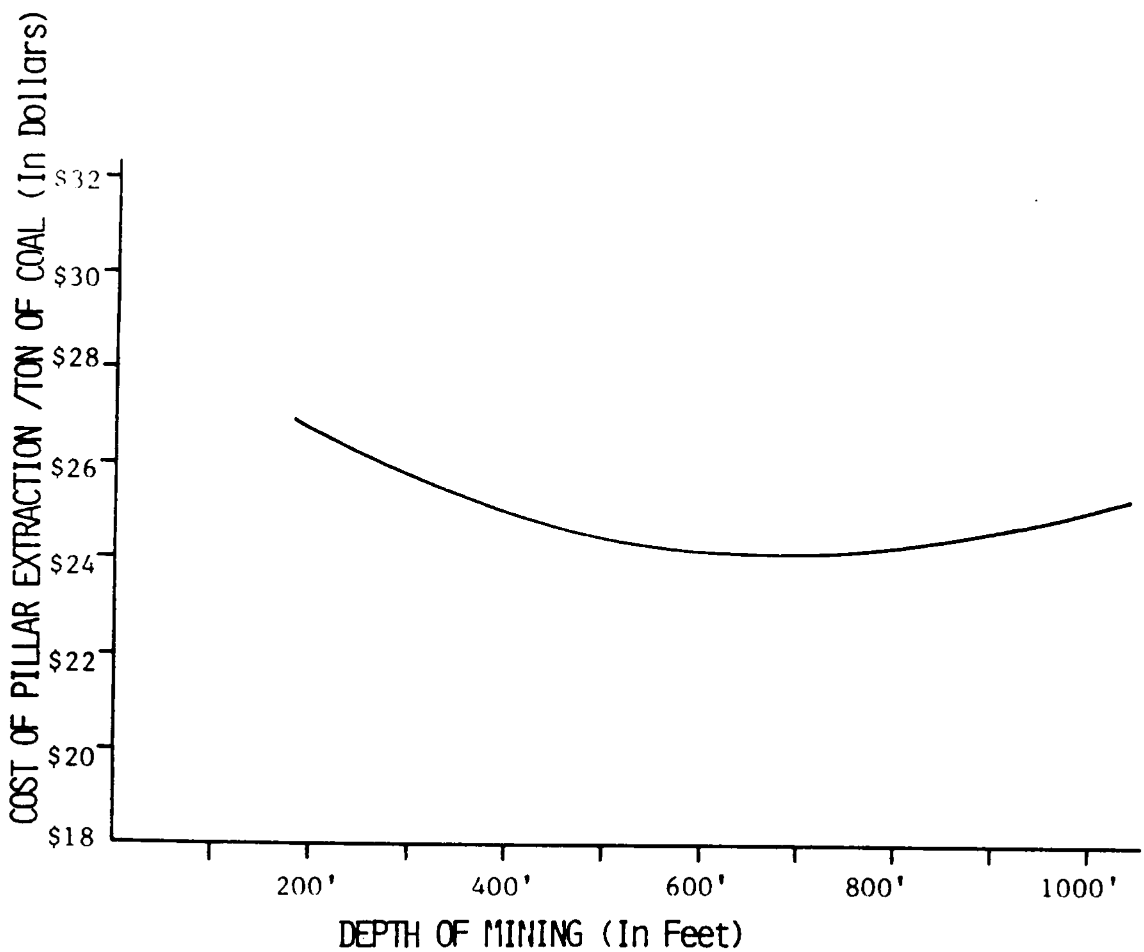
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION = 1.0



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MINING LIFE (DEVELOPMENT) = 20 Years
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 1.50
YEAR OF PRODUCTION = 1986
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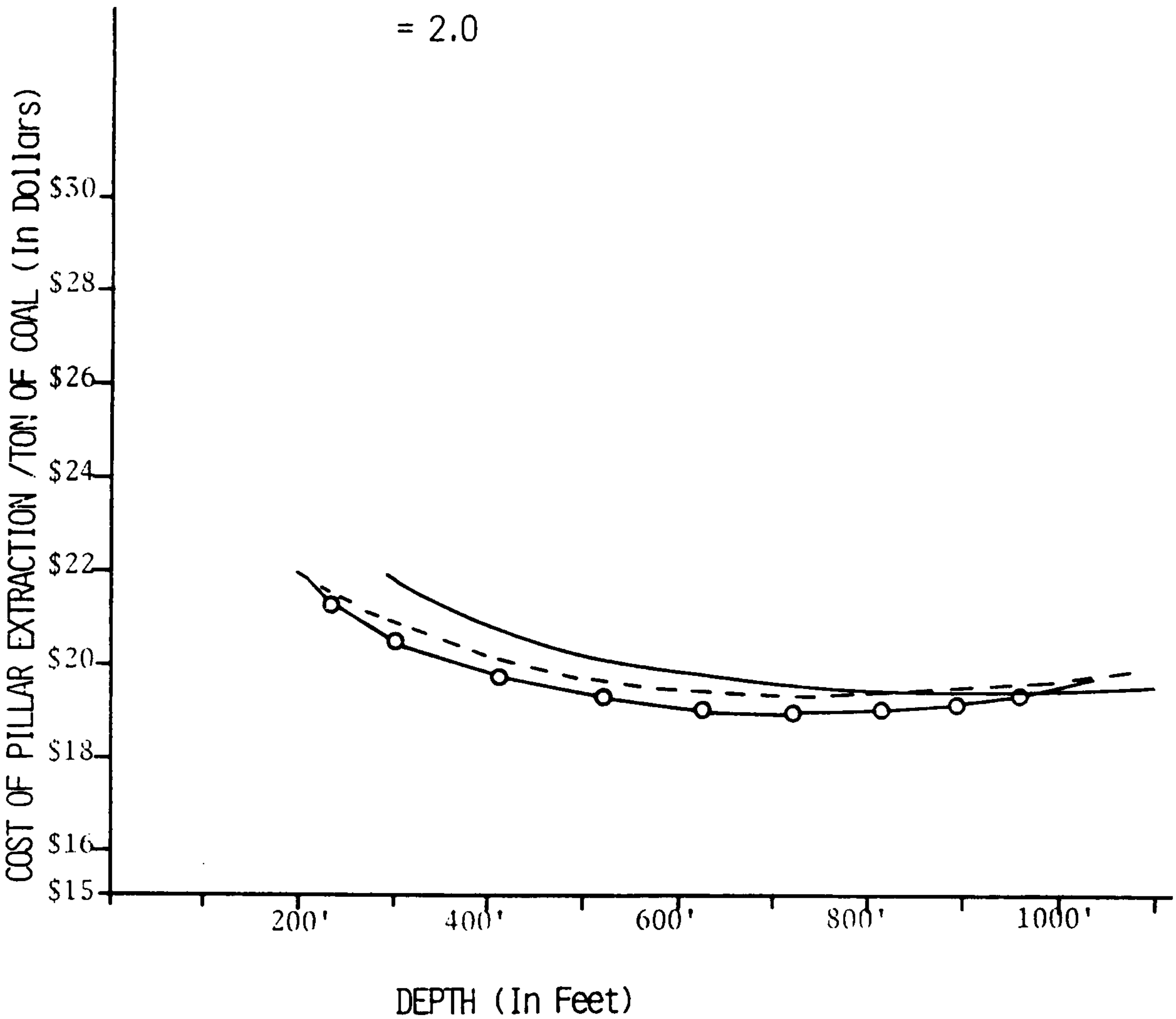
= 1.8
= 2.0



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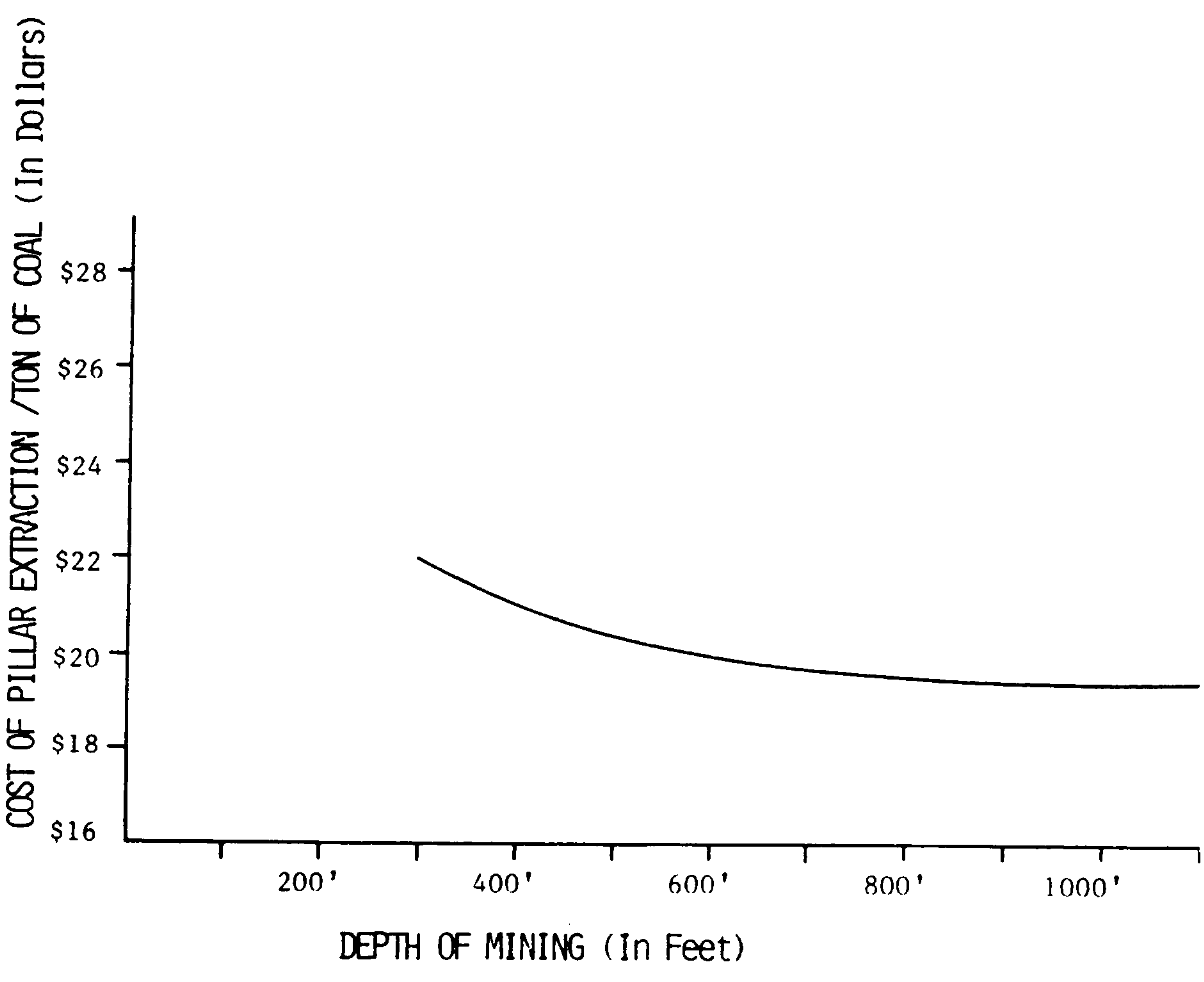
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 1 Million Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.5



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SUBSIDENCE COMPENSATION COST = 0 (Assumed)
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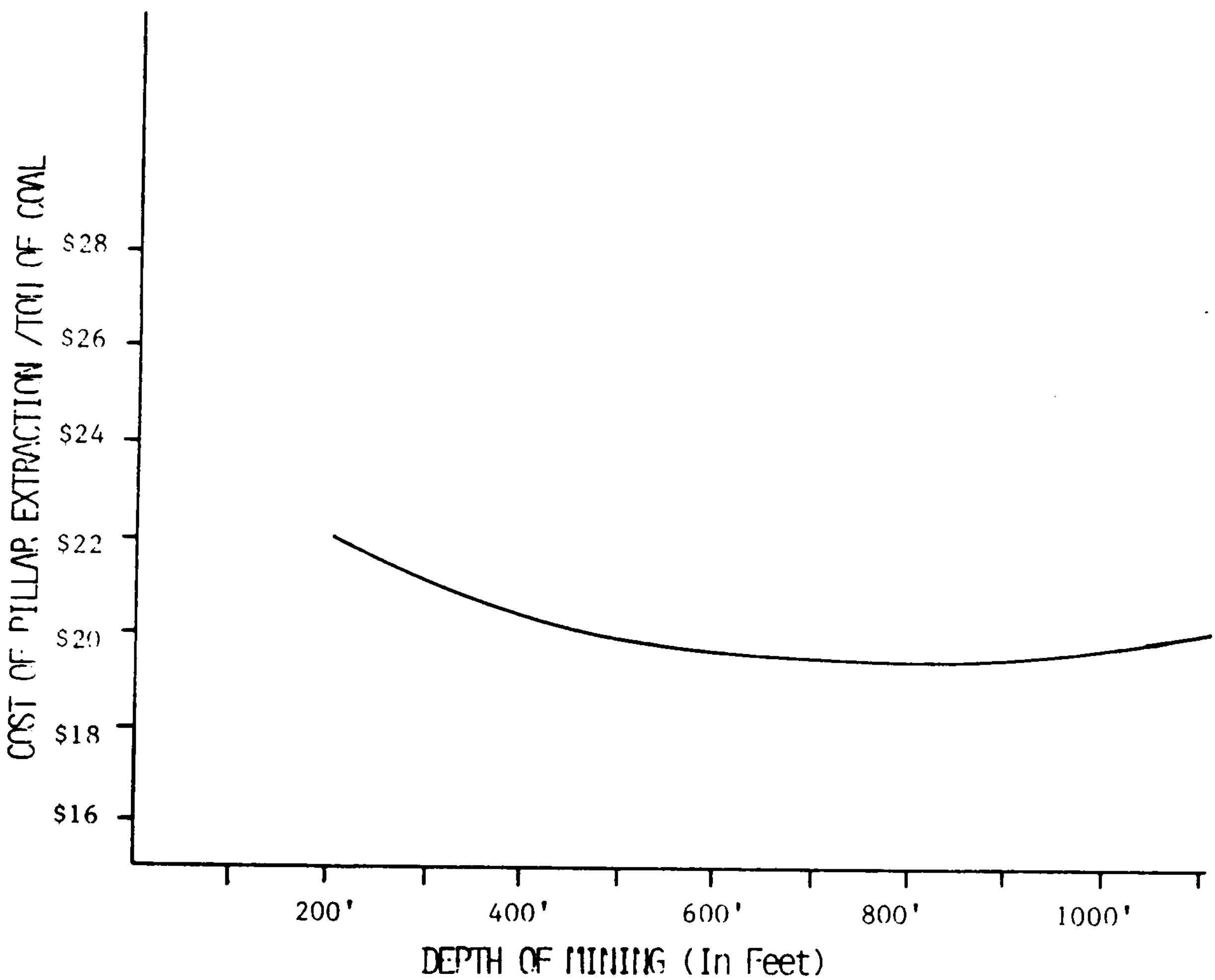
MINING LIFE = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.5



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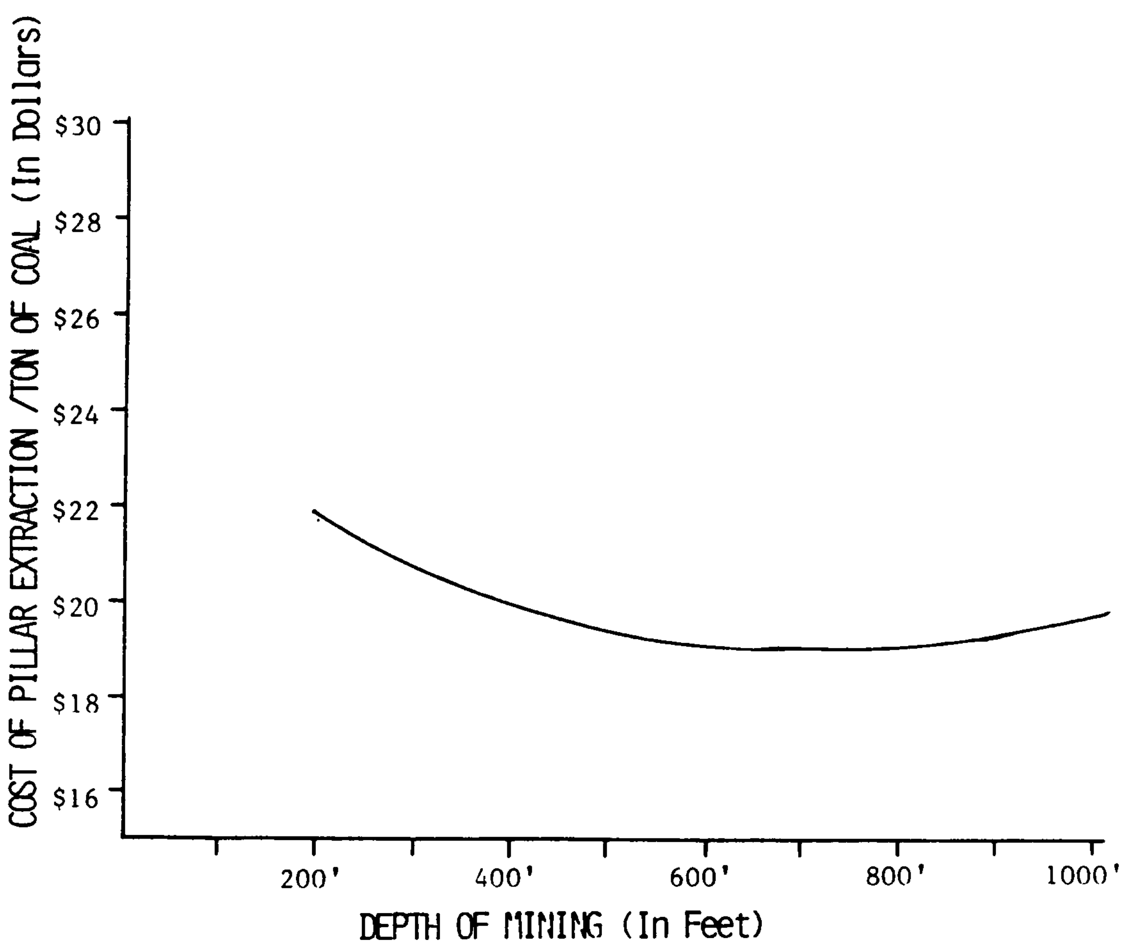
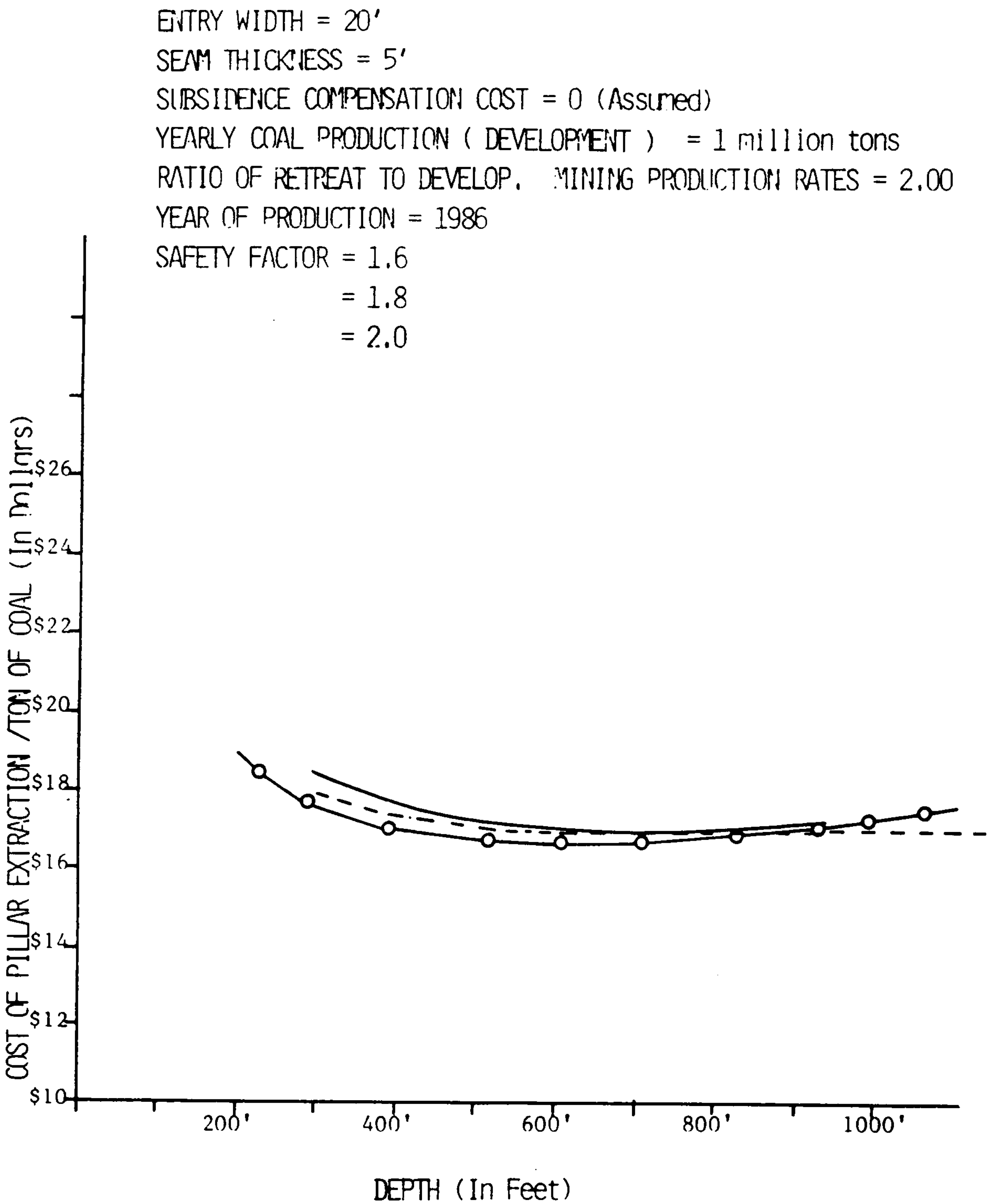
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 1.5

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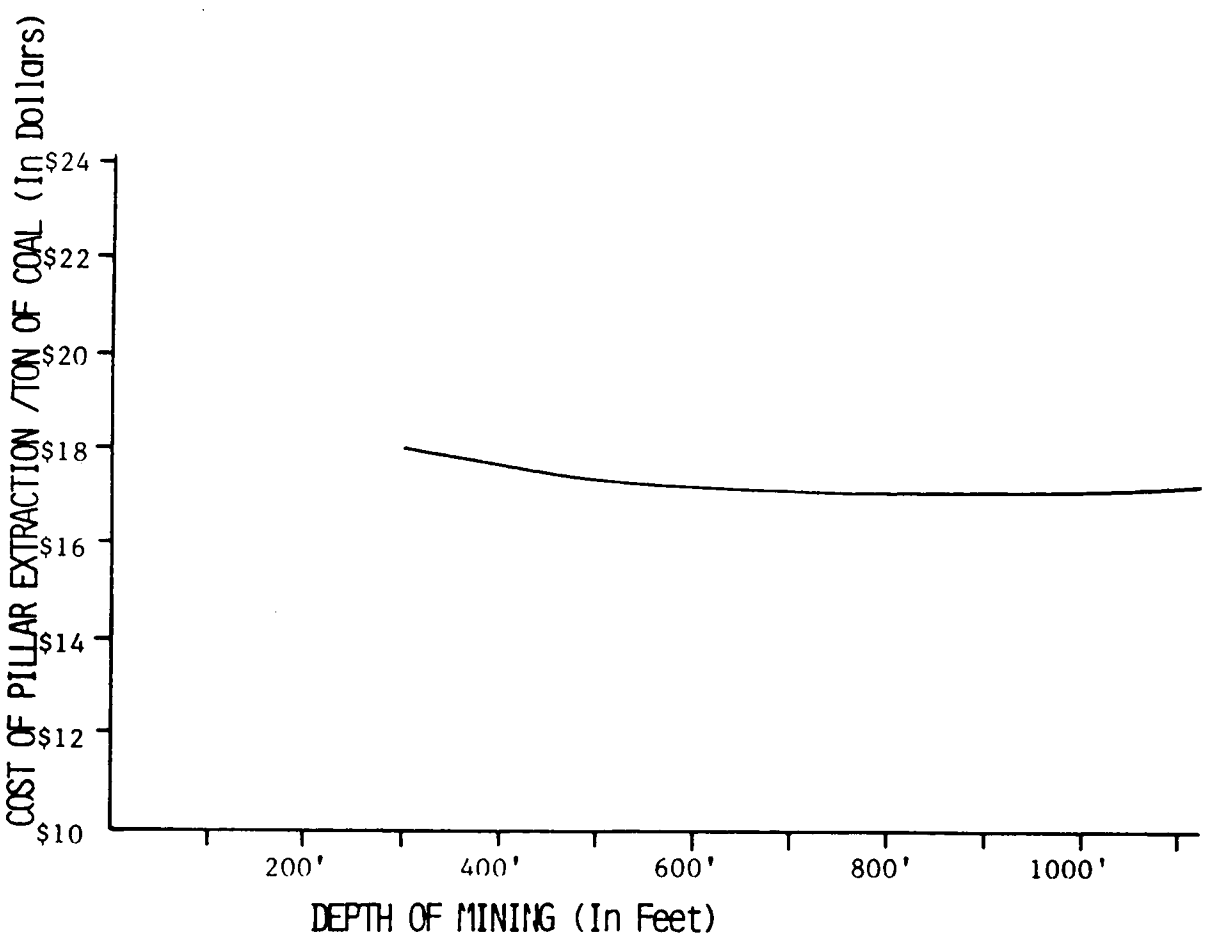
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATES = 2.00
YEAR OF PRODUCTION = 1986
SAFETY FACTOR = 1.6
= 1.8
= 2.0
COST OF PILLAR EXTRACTION /TON OF COAL (In Dollars)
$26
$24
$22
$20
$18
$16
$14
$12
$10
200'
400'
600'
800'
1000'
DEPTH (In Feet)

ENTRY WIDTH = 20'
SEAM THICKNESS = 5'
SAFETY FACTOR = 1.8 (Salamon's Equation)
SUBSIDENCE COMPENSATION COST = 0 (Assumed)
YEARLY COAL PRODUCTION (DEVELOPMENT) = 1 Million Tons
MINING LIFE (DEVELOPMENT) = 20 Years
YEAR OF PRODUCTION = 1986
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 2.0



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MINING LIFE (DEVELOPMENT) = 20 Years
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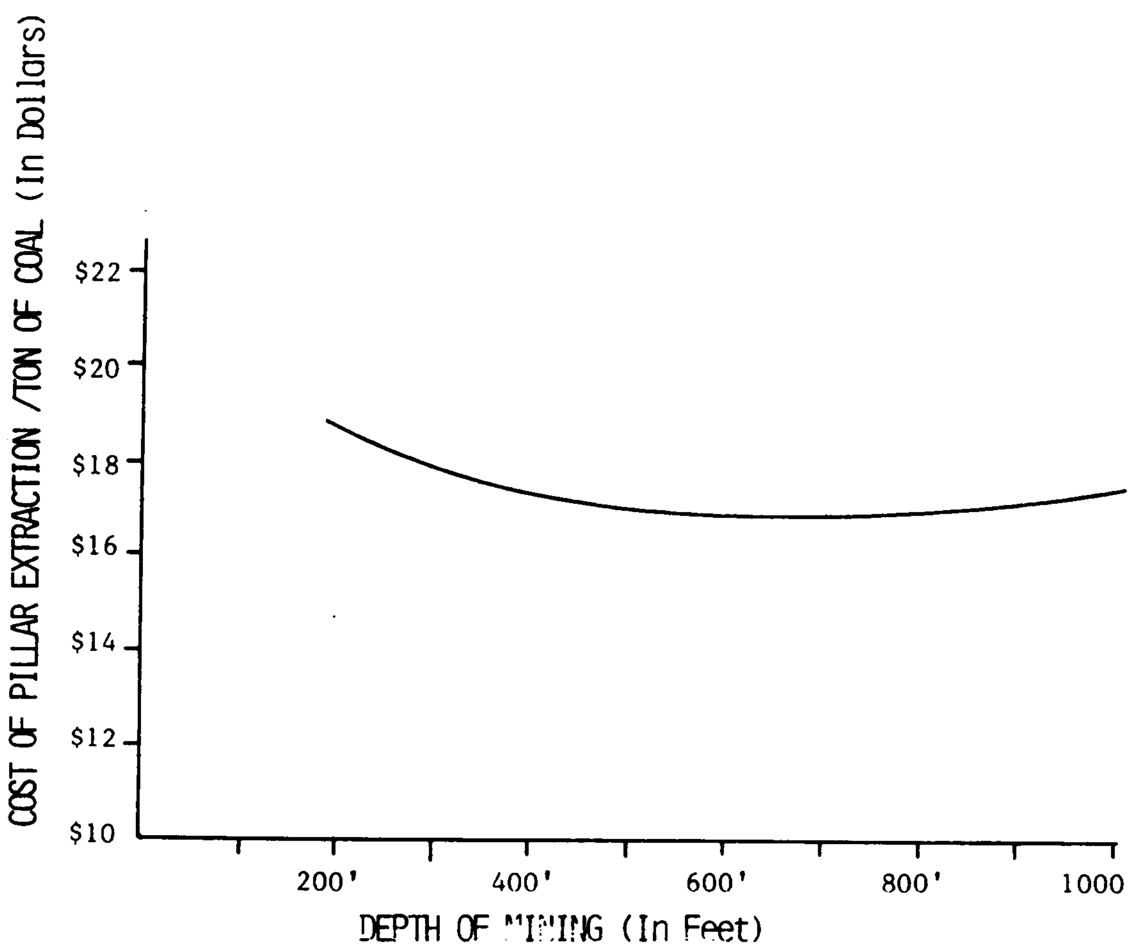
RATIO OF RETREAT TO DEVELOP. MINING PRODUCTION RATE = 2.0



APPENDIX F

PROBABILITY AND MONTE CARLO SIMULATION

F. PROBABILITY AND MONTE CARLO SIMULATION

Before using the concept of probability and Monte Carlo simulation for designing coal pillars it is required to test various coal samples and collect sufficient coal strength test data for the analysis.

F.1 Probability Distributions Useful in Generating Coal Strength

The properties of various probability distributions relevant to coal strength are presented as follows (Canmet, 1977):

Normal:

The best known and most frequently referenced random variable in the literature on random phenomena is the normal random variable (Schmidt, 1982). The probability density function of the normal random variable, Figure F. 1, is symmetric about its mean and is defined on the interval $(-\infty, \infty)$. The cumulative distribution function of the normal random variable cannot be expressed in closed form. The cumulative distribution function (c. d. f.) of the standard normal random variable may be used to determine values of the c. d. f. for a normal random variable, X, with mean μ and variance σ^2 by the transformation,

$$Z = \frac{X-\mu}{\sigma}$$

That is if X is normally distributed with mean μ and variance σ^2 then Z has a normal distribution with mean 0 and variance 1.

The following approximation for the c. d. f. of a normal random variable, X, with mean μ and variance σ^2 is used.

$$F(x) = \begin{cases} H(x), & \text{if } -\infty < x <= \mu \\ 1-H(x), & \text{if } \mu < x < \infty \end{cases}$$

where,

$$H(x) = (a_1t + a_2t^2 + a_3t^3) \cdot \frac{1}{\sigma\sqrt{2\pi}} \exp\{-\frac{(x-\mu)^2}{2\sigma^2}\}$$

$$t = \frac{1}{1+0.33267 \left|\frac{(x-\mu)}{\sigma}\right|}$$

$a_1 = 0.436186$, $a_2 = -0.1201676$, $a_3 = 0.9372980$

<u>Exponential</u>:

The exponential random variable is a member of the family of gamma random variables (Hadly and Whitin, 1973). The cumulative distribution function is:

$$F(x) = 1 - \exp[-\lambda x], \qquad 0 <= x < \infty$$

where,

λ is the parameter of the exponential distribution.

Uniform:

The uniform or rectangular random variable, Figure F.2, finds its application in Monte Carlo simulation (Fishman, 1973; and Naylor, 1973). The probability density function of the uniform random variable is constant over its range of definition and its cumulative distribution function is defined by:

$$F(x) = \frac{x-a}{b-a}, \qquad a <= x <= b.$$

Weibull:

The probability distribution functions of the Weibull random variable are depicted in Figure F.3. The cumulative distribution function of the Weibull random variable is given by:

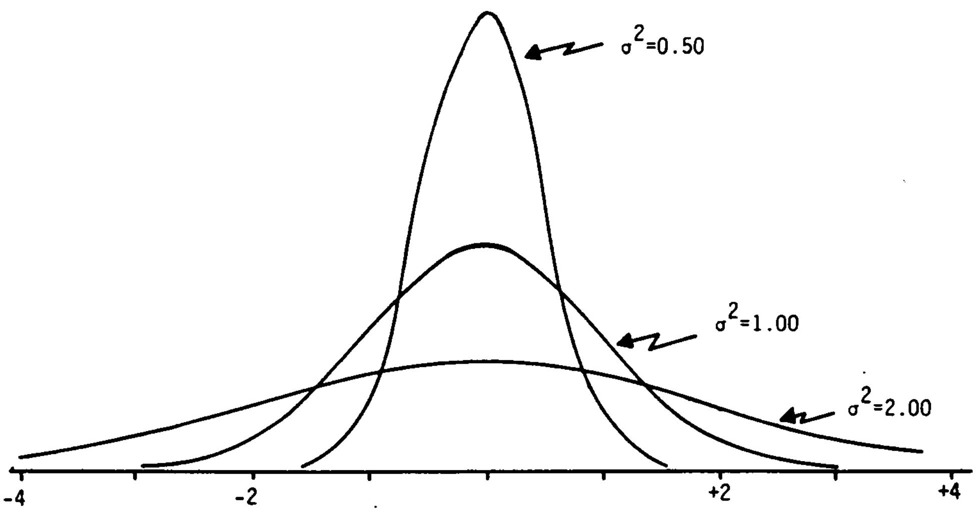
$$F(x) = 1 - \exp[-(x/b)^c], \qquad 0 <= x < \infty$$

where,

b, c are the parameters of the Weibull distribution.

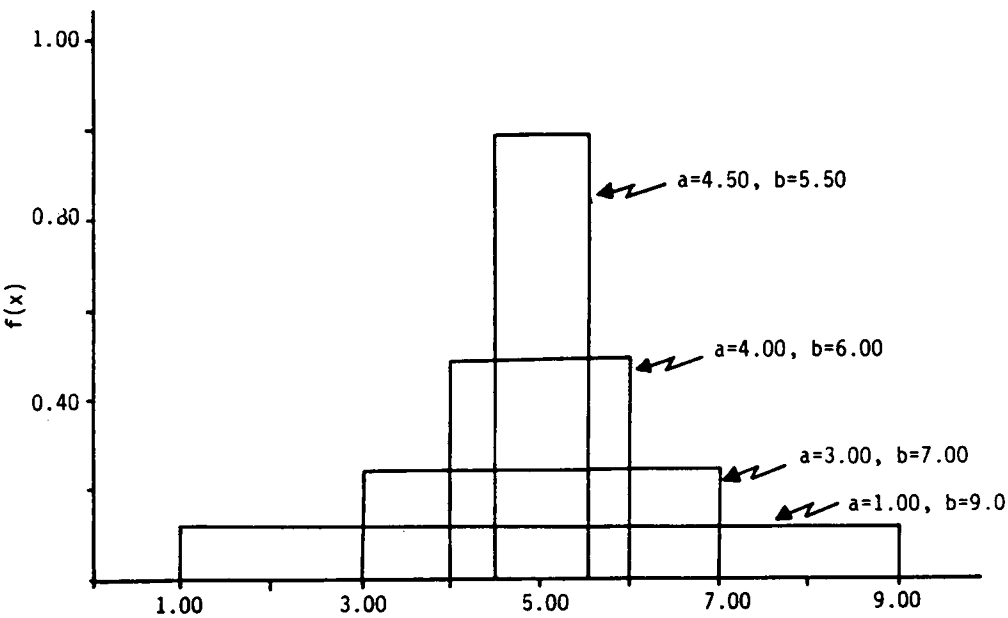
Log-Normal:

The lognormal random variable, Figure F.4, is functionally related to normal random variable. Specifically, if X is normally distributed with mean λ and variance β^2 then the natural logarithm of X, ln(X), has a log-normal distribution. The log-normal random variable is non-negative valued with probability density function,



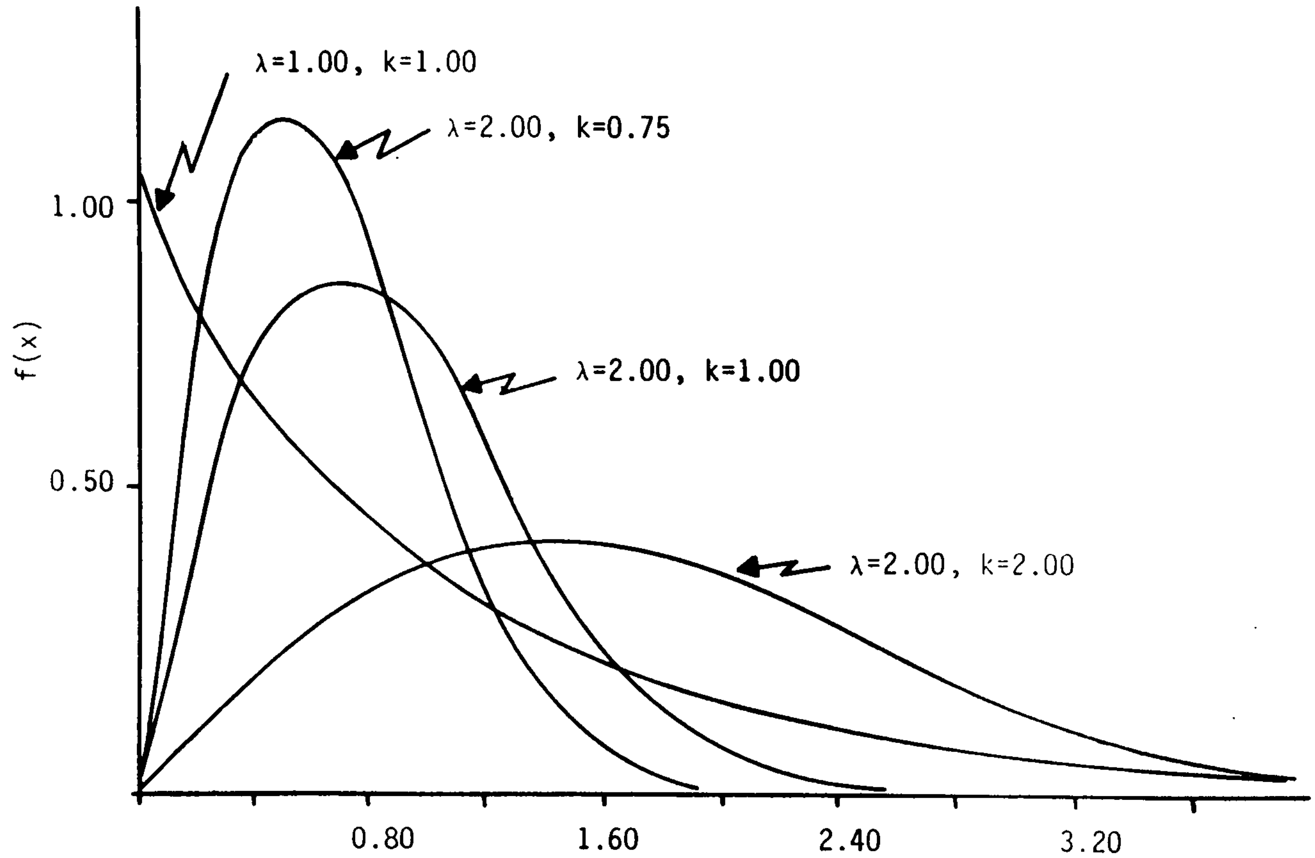
Probability Density Functions of the Normal Random Variable

Figure F.1

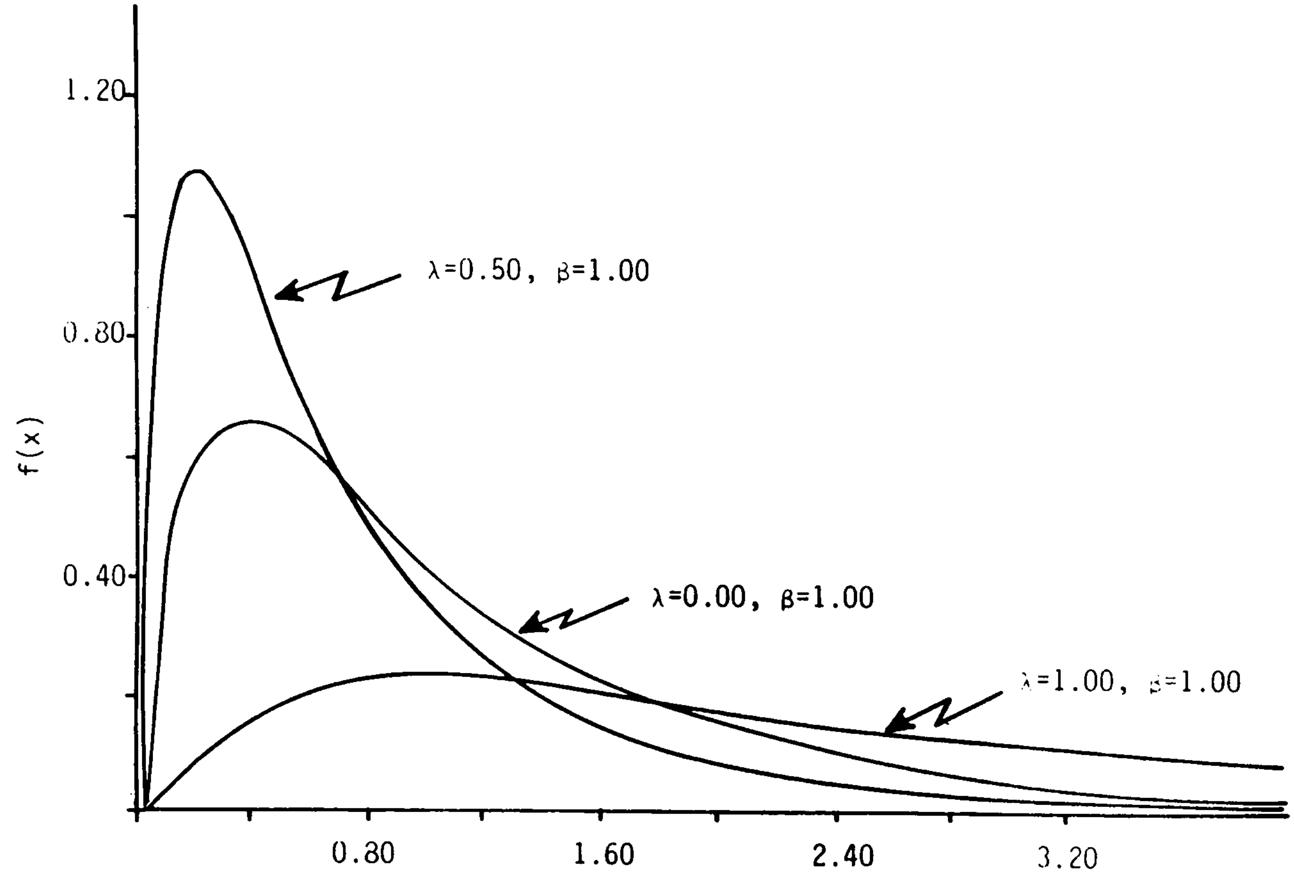


Probability Density Functions of the Uniform Random Variable
(Schmidt)

Figure F.2



Probability Density Functions of the Weibull Random Variable, (Schmidt, 1982) Figure F.3



Probability Density Functions of the Log-Normal Random Variable, (Schmidt, 1982) Figure F.4

which is always skewed to the right, Figure F.4 (Schmidt, 1982). Due to its relationship to the normal random variable, the cumulative distribution function of the log-normal random variable can be expressed in terms of the normal cumulative distribution function. If F(x) is the c. d. f. of the log-normal random variable, X, with parameters λ and β and G(q) the c. d. f. of the standard normal, Z, then

$$F(x) = G\left[\frac{\ln(x) - \lambda}{\beta}\right].$$

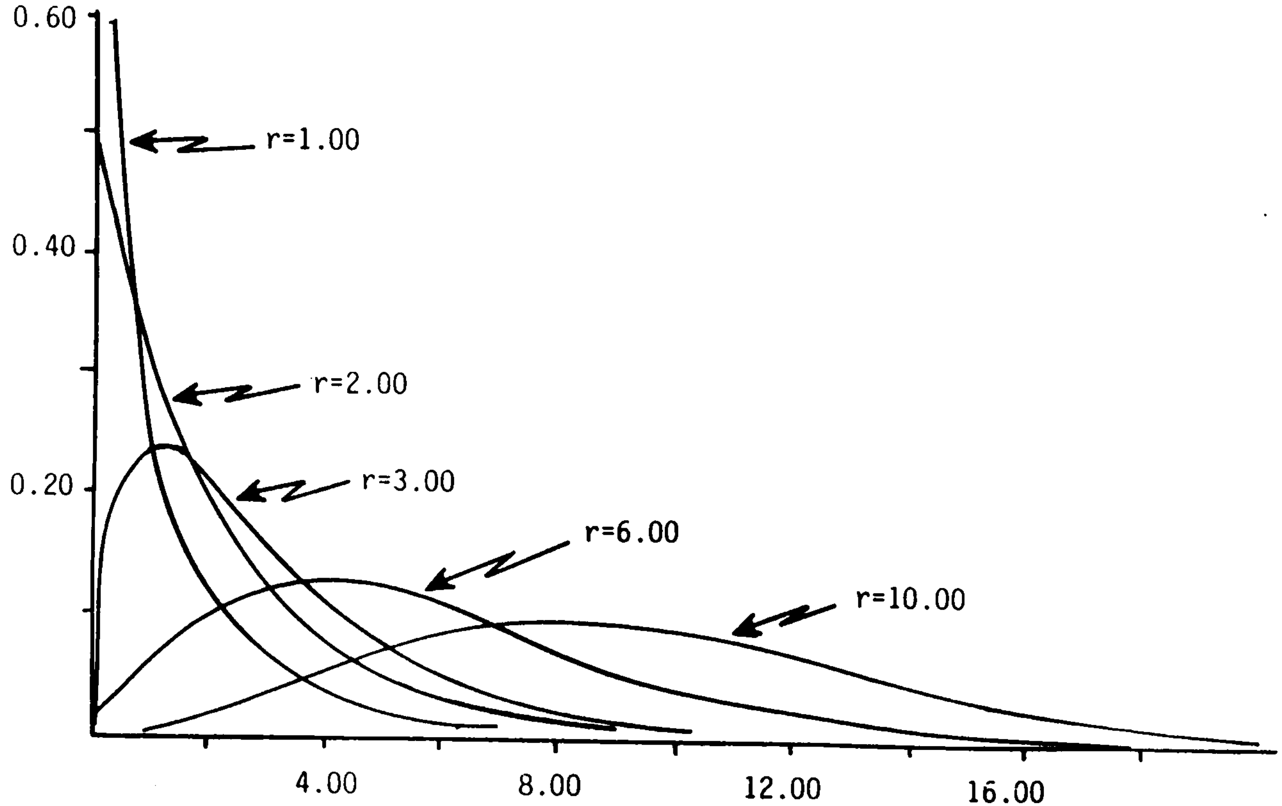
Hence, the c. d. f. of the log-normal random variable can be obtained from the equations discussed for c. d. f. of normal random variable.

<u>Chi-square, t and F</u>:

The Chi-square, t and F random variables are of importance in statistical methods and each is related to the normal (Schmidt, 1982). If $Z_1, Z_2, Z_3,$ ----, Z_n are n independent standard normal random variables and the random variable Y is defined by:

$$Y = \sum_{i=1}^{\gamma} Z_i^2$$

then Y has a Chi-square distribution with parameter γ called the degrees of freedom for the Chi-square random variable, Figure F. 5 . Now let Z be a standard normal



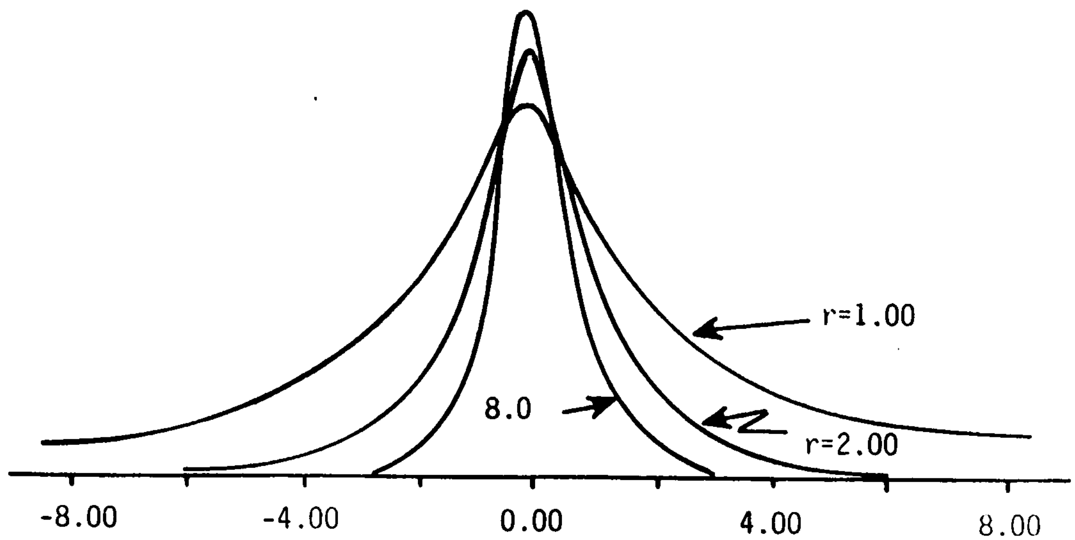
Probability Density Functions of the Chi-Square Random Variable,
(After Schmidt, 1982)
Figure F.5

random variable and Y a Chi-square random variable with degrees of freedom γ where Z and Y are independent. The random variable $Z/\sqrt{Y/\gamma}$ has a t distribution with γ degrees of freedom, Figure F.6. Finally, let X and Y be independent Chi-square random variables with γ_1 and γ_2 degrees of freedom respectively. The random variable $\frac{X/\gamma_1}{Y/\gamma_2}$ then has an F distribution with parameters γ_1 and γ_2, again called degrees of freedom, Figure F.7.

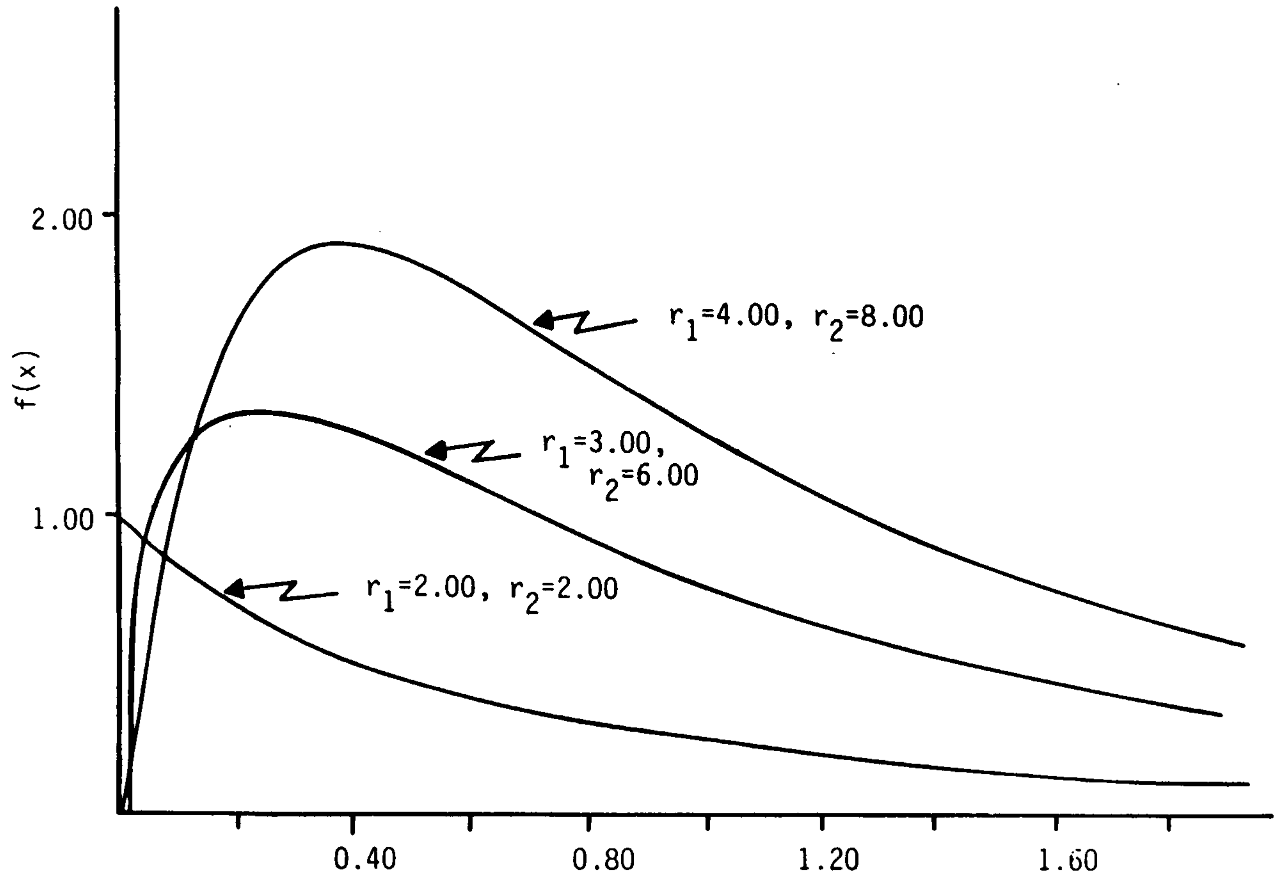
One of the most important uses of the Chi-square distribution is in the development of confidence intervals for the variance of a normal random variable and in hypothesis testing concerning a single variance (Schmidt, 1982). The Chi-square distribution is used to test the fit of a hypothesized probability distribution to observed data summarized in a frequency distribution (Johnson and Leone, 1964).

The application of t random variable is in developing confidence intervals for the mean of a normally distributed random variable when the true variance is unknown, and in testing hypotheses about the mean of such a population. It is similarly applied to the development of confidence intervals or tests of hypothesis concerning the difference of two means (Johnson and Leone, 1964; and Schmidt, 1982).

The F random variable is used for confidence interval estimation for the ratio of the variances of two normal



Probability Density Functions of the T Random Variable,
(After Schmidt, 1982)
Figure F.6



Probability Density Functions of the F Random Variable,

(After Schmidt, 1982) Figure F.7

random variables and in testing hypotheses concerning the ratio of variances (Johnson and Leone, 1964; and Schmidt, 1982).

Empirical Distribution:

If V_k is value of the k-th interval in the histogram and p_k is the corresponding cumulative probability, then (Schmidt, 1982):

$$p_k = p_{k-1} + m_{k-1}/N$$

where,

m_{k-1} = number of observations in the interval $[V_{k-1}, V_k)$

N = number of total observations.

It is assumed that $p_1 = 0$, $p_{n+1} = 1$ (n = number of intervals of the histogram).

F.2 Shape Characteristics of Probability Distribution:

The mean and variance of a random variable provide information regarding the central tendency and dispersion of a random variable. However, these measures offer little information as to the shape of the distribution. The shape of the distribution of interest is quite important in identifying the probability mass function or probability density function of the random variable (Schmidt, 1982).

The mode of a random variable is that value at which its probability mass function or probability density function achieves a relative maximum. It should be noted that a random variable may have several modes or points at which its distribution (p.m.f. or p.d.f.) "peaks". The median of a random variable is the smallest value which it may assume, m_e, such that:

$$P(X <= m_e) >= 0.50.$$

In other words m_e is that value of the random variable such that 50% of the area under the p.d.f. lies to the left of m_e and 50% to the right. However, such a point may not exist for a discrete random variable leading to the general definition given in the equation above.

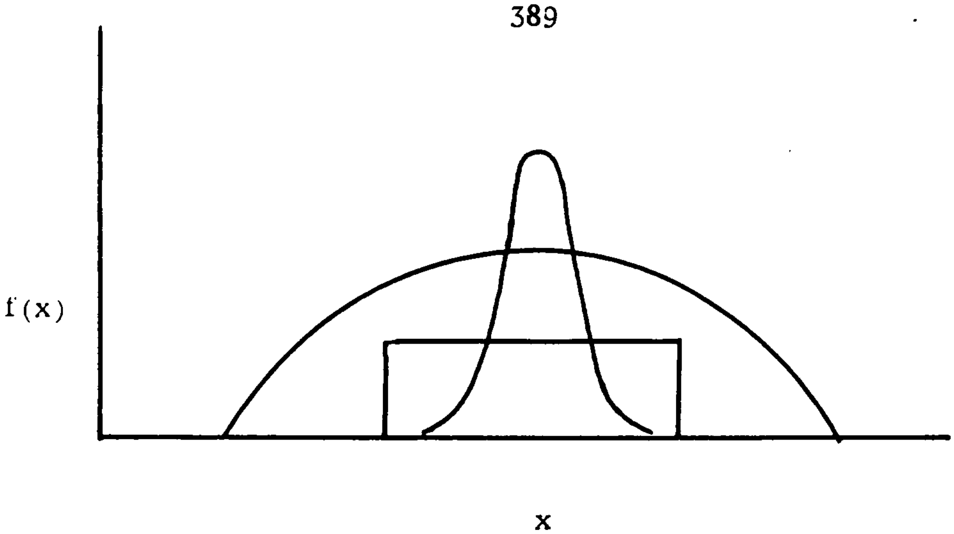
The p.m.f. or p.d.f. of a random variable with mean is symmetric if:

$$P(X <= \mu - e) = P(X <= \mu + e) \qquad \text{for all } e >= 0.$$

If the p.d.f. of a continuous random variable is symmetric and possesses a single mode, then its mean, mode and median are equal. Several symmetric distributions are illustrated in Figures F. 8 and F. 9.

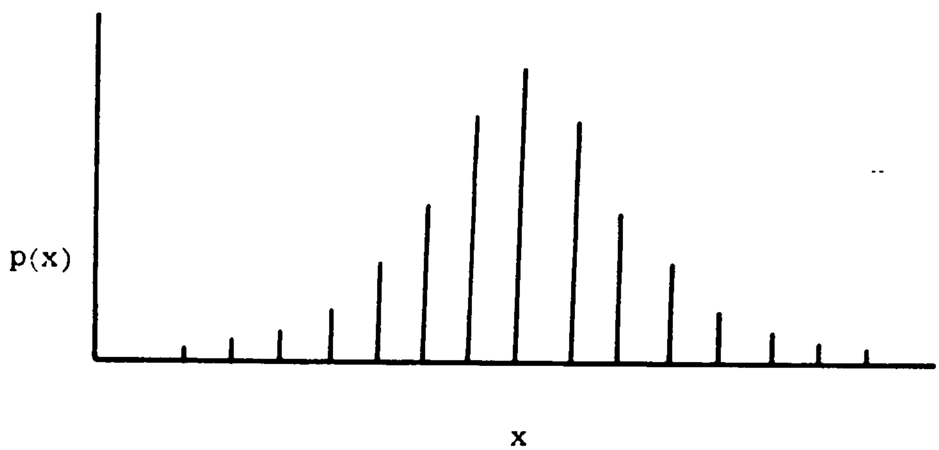
A p.m.f. or p.d.f. which is not symmetric is said to be skewed. Most distributions which are asymmetric are skewed to the right in the sense that they tend to "tail off" to the right as indicated in Figure F. 10. On the other hand, distributions which are skewed to the left have longer tails on the left than the right as shown in Figure F. 11.

The most frequently used measure of skewness of a



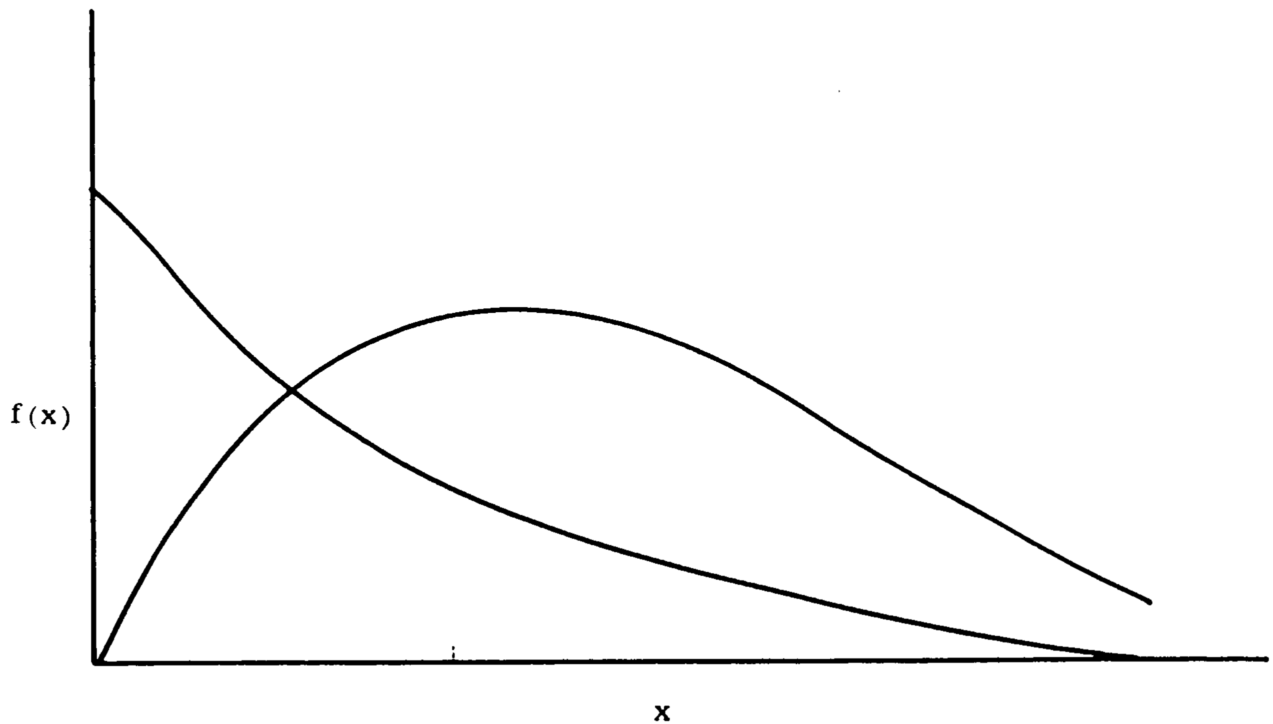
Symmetric Probability Density Function

Figure F.8



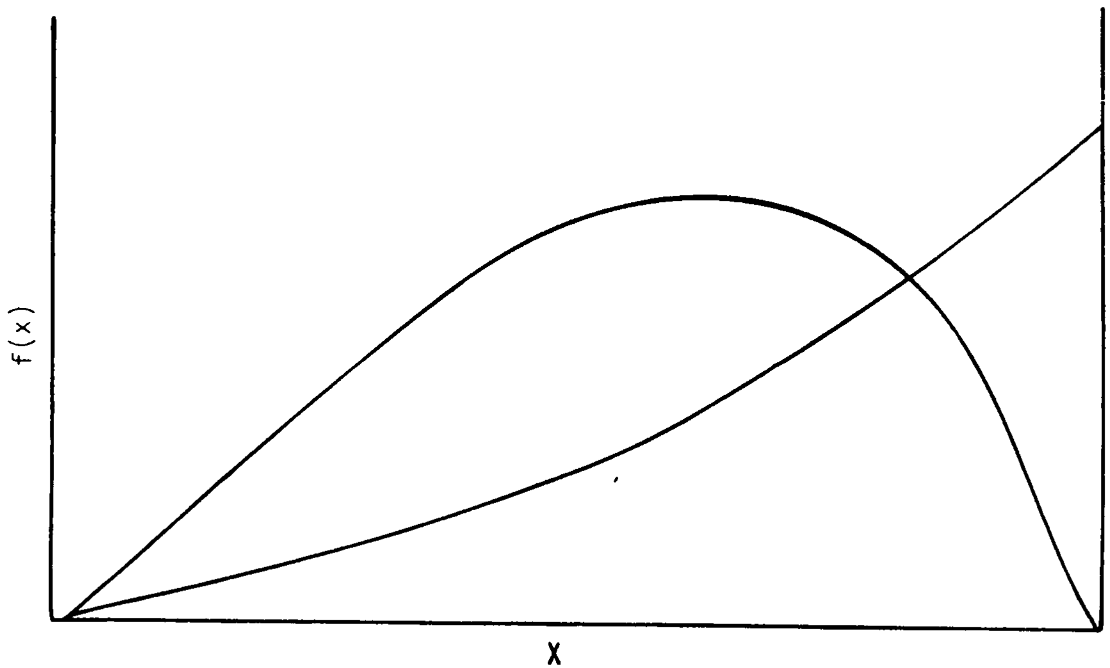
Symmetric Probability Mass Function, (Schmidt)

Figure F.9



Probability Density Functions Skewed to the Right, (After Schmidt)

Figure F.10



Probability Density Functions Skewed to the Left

Figure F.11

distribution is the coefficient of skewness given by:

$$\gamma_1 = \frac{\mu_3}{(\sigma^2)^{3/2}}$$

where,

μ_3 is the third central moment of X defined as

$$\mu_3 = E(X-\mu)^3.$$

and μ its mean and σ^2 is the variance of X. A positive value of γ_1 indicates that the distribution is skewed to the right while a negative value indicates skewness to the left. For symmetric distributions $\gamma_1 = 0$.

F.3 Identifying the Distribution of a Random Variable:

The most common way of identifying the type of distribution of a random variable is by plotting its frequency distribution in the form of a histogram and then comparing the shape of it with the contemplated theoretical distributions, finally selecting the one that looks most similar to the experimental shape. The shapes of various probability density functions are depicted in Figures F.1 to F.7 .

One can test more than one distribution for the same variable and select the one that results in the minimum Chi-square value. Finally, if it is not possible to fit the observed distribution into a known one, it is advisable to use an empirical distribution instead.

F.4 Estimation of the Parameters:

Estimation is the technique of developing suitable methods by which the observed data can be used to construct good estimates of the parameters present in a distribution. Ideally, the best estimation technique should be the one that yields the true value of the population parameters from the sample. However, such a technique does not exist with the current state of the art and the observed data, because a representation of the actual population will always give incomplete information about the true population. Thus, there is always some error in the estimation (Lindgren, 1976).

The observed data has been used to estimate the parameters of the various probability distributions:

(a) Uniform distribution: "a" is the minimum value observed, while "b" is the maximum.

(b) Normal distribution: "μ" is equal to mean sample value and "σ^2" the variance.

(c) Exponential distribution: "μ" is equal to $1/\lambda$ and the variance "σ^2" is equal to $1/\lambda^2$. The experimental estimators of μ and σ^2, that is $\bar{x}$ and s^2 can be used to calculate the value of λ.

(d) Empirical distribution: This has been discussed in the preceding sections.

F . 5 Testing the Probability Distribution:

In order to test whether the distribution of the data is in agreement with the selected theoretical distribution, a Chi-square test is performed. First, a histogram of the data points in a histogram cell is compared to the theoretically expected number. Finally, the value of Chi-square is found (Schmidt, 1982):

$$\chi^2 = \sum_{i=1}^{K} \left(\frac{f_i - E_i}{E_i}\right)^2 .$$

where,

f_i is the theoretically expected number of points in cell i.

E_i is the observed number of points in cell i.

k is the number of cells.

i = 1, 2, ----, k.

f_i is given by: $f_i = n\{F(V_{i+1}) - F(V_i)\}$

where:

n = number of data points.

$F(V_i)$ = value of the cumulative probability function for V_i.

V_i, V_{i+1} = lower and upper limit of cell i.

One should compare the computed values of χ^2 to the value found in statistical tables for k-m-1 degrees of freedom (m is the number of parameters of the distribution) and the desired level of significance. Generally, the experimental distribution is said to be in agreement with the theoretical one if the computed value of Chi-square is less than the one found in statistical tables.

F.6 <u>Generation of Random Variables</u>:

A random variable generator is a mechanism which produces values of a random variable. Ideally the variable values produced by a random variable generator should possess the same properties as the random variable simulated. The probability mass or density function of the generated random variable should be the same as that realized in the physical system. The values of the random variable generated should be truly random in the sense that they cannot be predicted with certainty (Rubinstein, 1981).

Random variables that are not independent are referred to as dependent or correlated random variables. The extent of the correlation or dependence between two random variables, X_1 and X_2, is usually measured by the coefficient of linear correlation, ρ, defined by:

$$\rho = \frac{\sigma_{12}}{\sigma_1 \sigma_2}$$

where σ_1 and σ_2 are the standard deviations of X_1 and X_2, respectively, and σ_{12} is the covariance between X_1 and X_2. If X_1 and X_2 have means μ_1 and μ_2, respectively, then the covariance between X_1 and X_2 is given by (Schmidt, 1982):

$$\sigma_{12} = E[(X_1 - \mu_1)(X_2 - \mu_2)].$$

If X_1 and X_2 happen to be independent random variables then

$$E[(X_1-\mu_1)(X_2-\mu_2)] = E(X_1-\mu_1)\ E(X_2-\mu_2) = 0$$

and $\rho = 0$.

The coefficient of linear correlation may assume values on the interval [-1, 1]. If $\rho = -1$, perfect negative correlation between X_1 and X_2 is said to exist in the sense that given the value of X_2, X_1 may be determined without error by a linear expression of form

$$X_2 = \alpha + \beta X_1 \qquad \text{where } \beta < 0.$$

For $\rho = 1$ perfect positive correlation exists and either X_1 or X_2 may be precisely determined given the value of the other and where $\beta > 0$.

If $0 < \rho < 1$, there is a tendency for the values of X_1 and X_2 to increase and decrease simultaneously, the tendency increasing with increasing ρ. The reverse is true if $-1<\rho<0$.

In summary, then, for the random variables X_1 and X_2:

1. $\rho \neq 0 \implies X_1$ & X_2 are dependent random variables.
2. $\rho = 0 \not\Longrightarrow X_1$ & X_2 are independent random variables.
3. Dependence of X_1 & $X_2 \not\Longrightarrow \rho \neq 0$

4. Independence of X_1 & X_2 ==> $\rho = 0$

Congruential Generator:

The most commonly used present-day method for generating pseudorandom numbers is one that produces a non-random sequence of numbers according to a recursive formula based on calculating the residues modulo of some integer m of a linear transformation. The numbers generated by the sequence appear to be uniformly distributed and statistically independent (Rubinstein, 1981).

$$X_{i+1} = (aX_i + c) \pmod{m}, \qquad i = 1, 2, ----, n,$$

where the multiplier a, the increment c, and the modulus m are non-negative integers. The module notation means that

$$X_{i+1} = aX_i + c - mk_i,$$

where $k_i = [(aX_i+c)/m]$ denotes the largest positive integer in $(aX_i+c)/m$.

Given an initial starting value X_o (also called the seed), the equation mentioned above yields a congruence relationship (modulom) for any value i of the sequence $\{X_i\}$. The random numbers on the unit interval (0, 1) can be obtained by:

$$U_i = \frac{X_i}{m}$$

where m is chosen as large as possible (Rubinstein, 1981).

c is relatively prime to m

$a \equiv 1 \pmod{g}$ for every prime factor g of m,

$a \equiv 1 \pmod{4}$ if m is a multiple of 4.

Inverse Transformation Method

This method which is most commonly used for generating random variates from different distributions is explained in detail here.

Let X be a random variable with cumulative probability distribution function (c.d.f.) F(x). Since F(x) is a nondecreasing function, the inverse function $F^{-1}(y)$ may be defined for any value of y between 0 and 1 as: $F^{-1}(y)$ is the smallest x satisfying F(x) >= y, that is,

$$F^{-1}(y) = \inf\{x:F(x) >= y\}, \qquad 0 <= y <= 1.$$

If U is uniformly distributed over the interval (0, 1), then

$$X = F^{-1}(U)$$

has cumulative distribution function F(x) (Rubinstein, 1981). So to get a value, say x, of a random variable X, obtain a value, say u, of a random variable U, compute $F^{-1}(u)$, and set it equal to x. A list of random variable generators has been depicted in Table F.1.

TABLE F.1

Random Variable Generators

Random Variable / Symbol	Range of Values	Parameter Restrictions	Random Variable Generator	Remarks
Normal $N(\mu,\sigma^2)$	$-\infty < X < \infty$	$-\infty < \mu < \infty$, $\sigma^2 > 0$	$X_1 = \mu + \sigma\sqrt{-2\ln(r_1)}\cos(2\pi r_2)$ $X_2 = \mu + \sigma\sqrt{-2\ln(r_1)}\sin(2\pi r_2)$	X_1 and X_2 are Independent
Exponential $E(\lambda)$	$0 \leq X < \infty$	$\lambda > 0$	$X = -1/\lambda \ln(r)$	
Weibull $W(k,\lambda)$	$0 \leq X < \infty$	$k > 0$, $\lambda > 0$	$X = k\{-\ln(r)\}^{1/\lambda}$	
Uniform $U(a,b)$	$a \leq X \leq b$	$-\infty < a < b < \infty$	$X = a + (b - a)r$	
Log-Normal $LN(\lambda,\beta)$	$0 \leq X < \infty$	$-\infty < \lambda < \infty$, $\beta > 0$	$X = e^y$	Y is $N(\lambda,\beta^2)$
Chi-Square $\chi^2(h)$	$0 \leq X < \infty$	even integer $h > 0$	$X = -2\ln\{\prod_{i=1}^{h/2} r_i\}$	
Chi-Square $\chi^2(h)$	$0 \leq X < \infty$	Integer $h > 0$	$X = \sum_{i=1}^{h} y_i^2$	$Y_1, Y_2, \ldots, Y_h$ are Independent $N(0,1)$
Chi-Square $\chi^2(h)$	$0 \leq X < \infty$	Odd Integer, $h > 0$	$X = 2\{-\ln(\prod_{i=1}^{m} r_i) + Y_1 Y_2\}$	$h = 2m + 1$, m=Integer Y_1 and Y_2 are Independent $B_e(0.5,0.5)$ and $E(1)$

TABLE F.1 -Continued

Random Variable Generators

Random Variable / Symbol	Range of Values	Parameter Restrictions	Random Variable Generator	Remarks
t $t(h)$	$-\infty < X < \infty$	Interger $h > 0$	$X = Y_1/\sqrt{Y_2/h}$	Y_1 is $N(0,1)$ and Y_2 and Independent $X^2(h)$
F $F(h_1, h_2)$	$0 \leq X < \infty$	Integer h_1 and h_2 $h_1 > 0$, $h_2 > 0$	$X = Y_1 h_2 / Y_2 h_2$	Y_1 and Y_2 are Independent $X^2(h_1)$ and $X^2(h_2)$

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GROUND CONTROL RAMIFICATIONS AND ECONOMIC IMPACT OF RETREAT MINING ON ROOM AND PILLAR COAL MINES

by

Arun Kumar

(ABSTRACT)

As the coal reserves at shallow depths become exhausted companies have to develop deeper deposits and increase percentage extraction to maintain production levels. Total extraction for room and pillar mines can only be achieved by pillar extraction. The unsupported roof increases during pillar extraction and hence the cost of ground control also increases. Nevertheless, pillar extraction where possible has many potential advantages such as decreased operating cost, increased utilization of reserves, and extended life of the mine. There are several variables such as depth, mining height, rock strength, mining geometry, roof and floor conditions, and retreat mining methods, which affect pillar extraction cost. Cost components of pillar extraction are classified as direct, indirect, fixed, and subsidence compensation costs. A discounted cash flow pillar extraction cost simulator has been developed and used to compute total pillar extraction cost for a variety of conditions and to explore the possibilities of optimizing ground control and retreat mining techniques to maximize extraction ratio. The computer program computes

the safe and optimum pillar dimensions and determines the suitable pillar extraction method for the computed pillar width. Pillar extraction cost components are generated and totalled using the net present value method by the simulator. The total extraction cost simulator evaluates the potential advantages of pillar extraction and tests individual variables for sensitivity to changes in other variables attributable to ground control and pillar extraction techniques. Cost of pillar extraction per ton of coal versus depth is presented in the form of a simple nomogram by the simulator. The simulator can be used to determine the economic feasibility of pillar extraction at a particular depth, geologic and mining environment when the market price of mined coal is known.